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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXVII.

FEBRUARY, 1897, TO JULY, 1897.

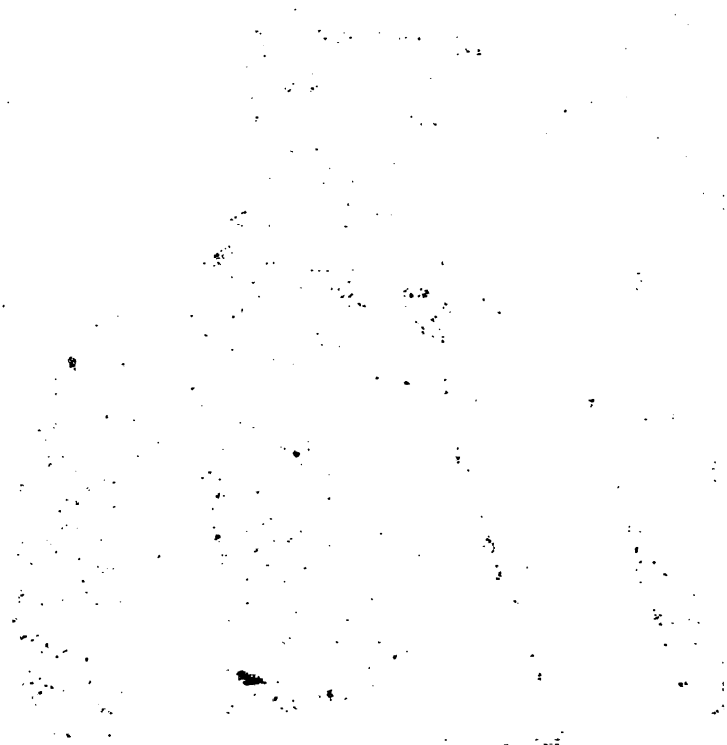
INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1898.

J. F. HOLLOWAY.

DIED SEPTEMBER 2D, 1896.

(See Biographical Notice, vol. xxvi., p. 827.)



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PHILADELPHIA:

SHERMAN & CO., PRINTERS.

PREFACE.

At the close of the present volume several discussions will be found, which, having been formally presented at the Atlantic City Meeting, February, 1898, strictly belong to the succeeding volume, but are included here in order not to separate them too far from other papers and discussions on the subjects of which they treat.

The excellent portrait of Mr. J. F. Holloway, introduced as a frontispiece to this volume, should have accompanied the Biographical Notice of Mr. Holloway in Vol. XXVI.

R. W. R.

July, 1898.

SEP 3 1898

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For year ending February, 1898.*

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* The following officers were elected at the Annual Meeting, February, 1898: *President*, C. Kirchhoff, New York City; *Vice-Presidents* (to serve two years), E. D. Peters, Jr., Dorchester, Mass.; A. R. Ledoux, New York City; Leon P. Feustman, Mexico City, Mexico; *Managers* (to serve three years), R. P. Rothwell, New York City; W. J. Olcott, Duluth, Minn.; W. B. Devereux, Glenwood Springs, Colo.; *Treasurer*, Theodore D. Rand, Philadelphia, Pa.; *Secretary*, Rossiter W. Raymond, New York City.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO FEBRUARY, 1898.

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XVII.	Philadelphia, Pa.,†	June, 1876,	v.	3
XVIII.	Philadelphia, Pa.,	October, 1876,	v.	19
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* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

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* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida.

PUBLICATIONS.

THE publications of the Institute comprise:

PAMPHLETS.

1. The minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed without charge to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows:

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RULES

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I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote, and members or associates whose post-office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance on the first day of each calendar year. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary, and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may he transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to

the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members) a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Other meetings shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS AND PUBLICATIONS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute shall

be printed in the *Transactions*. Intimation, when practical, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers. The published papers and volumes of *Transactions* shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the Rules.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade; nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute, as provided in these Rules; *Provided*, however, that committees may be appointed by the Council or the Institute to make investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as Scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these Rules from and after the publication of said announcement by the Secretary.

Proceedings of the Seventy-Second (Twenty-Seventh Annual) Meeting, Chicago, Ill., February, 1897.

General Committee.

Robert W. Hunt, *Chairman;* James F. Lewis, *Secretary;* Arthur V. Abbott, Samuel L. Brown, George B. Bartlett, J. C. Bartlett, Alfred S. Bertolet, Charles E. Billin, Thomas S. Blair, Edward L. Brown, M. C. Bullock, Charles S. Burt, N. L. Burwell, W. S. Calhoun, John S. Cary, William J. Chalmers, O. Chanute, Howard F. Chappell, Samuel S. Chisholm, Townsend V. Church, F. W. Clarke, E. A. S. Clarke, John Cliff, Walter G. Coolidge, F. K. Copeland, George M. Davidson, Fred A. Delano, Henry P. Dickinson, Richard D. Divine, William D. Ewart, Percy L. Fearn, C. H. Ferry, A. W. Fiero, Charles H. Foote, F. H. Foote, Robert Forsyth, George C. Gardner, James P. Gardner, John W. Gates, F. A. Gritzner, E. Lee Heidenreich, Hiero B. Herr, H. L. Hollis, L. Holmboe, William Hoskins, C. F. Howe, Edward C. Hegeler, Julius W. Hegeler, Edward E. Jewell, Frank Julian, John S. Kellogg, F. W. Matthiessen, Robert J. McClure, C. H. McCullough, Squire H. Macomber, John W. Meier, George Merryweather, Charles L. Miller, E. Kennard Mitting, Louis Mohr, George S. Morison, Jay C. Morse, Prof. Milton Moss, Frank Moynau, George P. Nichols, George S. Oliver, F. C. Osgood, John C. Parke, H. W. Parkhurst, Orrin B. Peck, Frederick Pelouze, Edward W. Penfield, Edward C. Potter, F. H. Prentiss, F. B. F. Rhodes, Morris Sellers, F. J. v. Skiff, H. S. Smith, Michael Smith, General William Sooy Smith, John L. Souther, W. R. Stirling, Bertrand S. Summers, William P. Schwartz, G. A. Trube, Jacob T. Wainwright, W. R. Walker, J. C. Walker, Reno B. Wellman, George D. Whitcomb, Jasper Whiting, Bruce Clarke White, Albert H. Wolf, Thomas W. Yardley.

Special Committees.

Finance.—Hiero B. Herr, *Chairman;* W. J. Chalmers, George Merryweather, Louis Mohr, O. Chanute, F. K. Copeland, Robert Forsyth, H. S. Smith.

Reception.—General William Sooy Smith, *Chairman;* John C. Parkes, A. W. Fiero, George S. Morison, M. C. Bullock, Edward C. Potter, T. W. Yardley.

Entertainments and Excursions.—F. J. v. Skiff, *Chairman;* George M. Davidson, Fred A. Delano, W. R. Walker, Charles H. Foote, H. M. Parkhurst, George P. Nichols, Charles E. Billin.

Hotels and Headquarters.—H. L. Hollis, *Chairman;* William Hoskins, Bertrand S. Summers.

Banquet.—Charles E. Billin, *Chairman;* E. A. S. Clarke, Robert W. Hunt, J. F. Lewis.

Music.—Charles E. Schauffler.

Ladies' Committee.

Mesdames William J. Chalmers, Hiero B. Herr, George Merryweather, Louis Mohr, O. Chanute, Robert W. Hunt, F. K. Copeland, William Sooy Smith, John

C. Parkes, A. W. Fiero, H. S. Smith, James F. Lewis, M. C. Bullock, Edward C. Potter, F. J. v. Skiff, George M. Davidson, W. R. Walker, Charles H. Foote, H. W. Parkhurst, George P. Nichols, H. L. Hollis, William Hoskins, Charles E. Billin, E. A. S. Clarke.

Hotel Headquarters.—The Auditorium, in which also all sessions after the first were held.

The opening session was held on Tuesday evening, February 16th, at Kimball Hall, corner of Wabash Avenue and Jackson Street. Mr. Robert W. Hunt, Chairman of the General Reception Committee, made an address of welcome, which was responded to by President Spilsbury.

Mr. James F. Lewis, Secretary of the General Reception Committee, presented cordial offers of hospitality from the Western Society of Engineers, the Armour Institute of Technology, the University of Chicago, the Chicago Academy of Sciences and the Art Institute.

The President announced the appointment of Messrs. H. P. Bellinger, James W. Abbott and W. J. Taylor, as Scrutineers, to examine the ballots received for officers and report the result at a later session.

In the absence of the author, the Secretary read a Biographical Notice of Joseph D. Weeks, by A. E. Hunt, Pittsburgh, Pa.

The following paper was read by the author:

Biographical Notice of Alexander Trippel, by R. W. Raymond, New York City.

The Secretary mentioned the recent death of Mr. George W. Goetz, and announced a Biographical Notice, to be published hereafter, by Nelson P. Hulst, Milwaukee, Wis., and S. T. Wellman, Cleveland, O.

The following paper was read and illustrated with numerous lantern-views:

The Chicago Main Drainage Channel, by James F. Lewis, Chicago, Ill.

The second session was held Wednesday morning, February 17th, in the Auditorium Hotel, President Spilsbury in the chair. The following papers were read and discussed:

The Calorific Value of Certain Coals as Determined by the Mahler Calorimeter, by N. W. Lord and F. Haas, Columbus, O. Read by Professor Lord.

Notes on the Determination of Insoluble Phosphorus in Iron Ores, by Charles T. Mixer and Howard W. DuBois, Philadelphia, Pa. Read by the Secretary in the absence of the authors.

The Quality of the Boiler-Water Supply of a Portion of Northern Illinois, by J. A. Carney, Aurora, Ill. Read by the author.

The following paper was read by the author, and illustrated with lantern-views.

Methods of Iron-Mining in Northern Minnesota, by Prof. F. W. Denton, Minneapolis, Minn.

The third session was held Wednesday afternoon at the Auditorium Hotel, President Spilsbury in the chair. The following paper was read and discussed:

The Handling of Material at the Blast-Furnace, by Axel Sahlin, Sparrow's Point, Md.

Mr. R. W. Hunt, Chicago, Ill., read a Brief Note on Rail Specifications.

The discussion of Mr. Sauveur's paper (Colorado meeting) on "The Microstructure of Steel and Current Theories of Hardening" was then begun, and the Secretary presented numerous contributions in print and manuscript, which were followed by others from members present.

The fourth and final session was held Thursday morning, February 18th, in the Auditorium Hotel, President Spilsbury in the chair. The following papers and contributions were presented by the Secretary in the absence of the authors:

The Distribution of the Precious Metals and Impurities in Copper, and Suggestions for a Rational Mode of Sampling, by Edward Keller, Baltimore, Md.

Sulphur in Embreville Pig-Iron, by Guy R. Johnson, Embreville, Tenn.

Discussion of Professor Christy's paper on the Solution and Precipitation of the Cyanide of Gold, by Alfred James, Glasgow, Scotland, E. B. Wilson, Salem, Va., and others.

The Hand-Auger and Hand-Drill in Prospecting-Work, by Charles Catlett, Staunton, Va.

The Spitzkasten and Settling-Tank, by R. H. Richards and C. E. Locke, Boston, Mass.

The Manganese-Deposits of the Department of Panama, Republic of Colombia, by E. J. Chibas, New York City.

The Precipitation of Gold by Zinc-Thread from Dilute and Foul Cyanide Solutions, by Alfred James, Glasgow, Scotland.

The Geology of the Magnetites near Port Henry, N. Y., and Especially Those of Mineville, by J. F. Kemp, New York City.

The following papers were read by title:

Sorting Before Sizing, by R. H. Richards, Boston, Mass.

The Genesis of Certain Auriferous Lodes, by Dr. John R. Don, Dunedin, New Zealand.

The Cement-Materials of Southwest Arkansas, by J. C. Branner, Stanford University, Cal.

The Chromite-Deposits on Port au Port Bay, Newfoundland, by G. W. Maynard, New York City.

Notes on the Northern Black Hills of South Dakota, by Dr. Persifor Frazer, Philadelphia, Pa.

A Decimal Gauge for Wire and Sheet-Iron, being an Abstract of the Report of Mr. Albert Ladd Colby to the Association of American Steel Manufacturers, by R. W. Raymond, New York City.

The Scrutineers reported the following persons to have been elected as officers of the Institute:

PRESIDENT.

Thomas M. Drown, ...	South Bethlehem, Pa.

VICE-PRESIDENTS.

(To serve two years.)

D. W. Brunton, ...	Aspen, Colo.
W. E. C. Eustis, ...	Boston, Mass.
James Douglas, ...	New York City.

MANAGERS.

(To serve three years.)

C. W. Goodale, ...	Butte, Mont.
Frank Lyman, ...	Brooklyn, N. Y.
Frank McM. Stanton, ...	Houghton, Mich.

TREASURER.

THEODORE D. RAND, ... Philadelphia, Pa.

SECRETARY.

ROSSITER W. RAYMOND, ... New York City.

The annual report of the Council was presented, as follows:

ANNUAL REPORT OF THE COUNCIL.

In accordance with the rules the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer shows receipts from all sources for the eleven months ending December 31st (including $5176.62 on hand at the beginning of the year) of $32,411.27, and expenditures of $23,689.51, leaving a surplus of $8721.76, being a gain upon the surplus of February 1, 1896, of $3545.15. In addition to this, the Treasurer holds United States bonds of the par value of $2900 and a special deposit of $4598, proceeds of United States bonds called in and paid by the government, which fund has not been permanently reinvested. The detailed statement of receipts and expenditures is as follows:

RECEIPTS.

Balance from statement, February 1, 1896, . .		$5,176 62
Annual dues, ...	$20,903 13	
Life-memberships, ...	2,176 74	
Binding of *Transactions*, ...	1,760 78	
Sale of publications, ...	1,945 75	
Electrotypes, ...	17 10	
Interest on United States bonds and deposits, .	431 15	
		$27,234 65
		$32,411 27

DISBURSEMENTS.

Printing volume XXV. *Transactions*, . . .	$2,902 17
" pamphlet edition of papers, . . .	3,140 05
" mailing list, ...	120 00
" circulars and ballots, ...	215 93
Binding volume XXV. and miscellaneous volumes of *Transactions*, ...	1,488 97
" exchanges, ...	241 24
Engraving and electrotyping, ...	1,032 00
Postage, including post-office box-rent, . .	682 14
Stationery, ...	209 71
Rent (nine months), ...	600 00
Express and freight charges, ...	2,038 70
Carried forward, ...	$12,670 91

Brought forward,	$12,670 91	
Telephone,	91 25	
Telegrams, cablegrams and car fare,	18 14	
Coal, ice and gas,	45 65	
Salaries, including clerks, stenographers and expenses of editing, and proof-reading,	9,694 50	
Storage of *Transactions*,	100 92	
Special stenographers and expenses of meetings,	714 78	
Office supplies and repairs,	283 95	
Insurance,	59 40	
Miscellaneous,	10 01	
		$23,689 51
Balance,		8,721 76
		$32,411 27

The surplus now on hand will enable the Council to increase its reserved fund to an amount equivalent to $100 for each living life-member; and investments sufficient for this purpose have been authorized.

Two meetings were held during the year, the annual meeting in February at Pittsburgh and a Colorado meeting in September. Accounts of these meetings have been already received by members in the form of the printed "Proceedings," which furnish abundant evidence of their successful and enjoyable character.

Changes in membership have taken place during the year as follows: 168 members and 17 associates have been elected; 12 associates have become members; the deaths of 1 honorary member, 36 members and 1 associate have been reported; 36 members and 7 associates have resigned; and 93 members and 6 associates have been dropped for continued default in the payment of dues. These changes are tabulated as follows, showing a net gain of 5 in total membership, which is largely increased by the number of applications already approved for election at the annual meeting now pending:

	H. M.	F. M.	M.	A.	Totals.
At date of last report	13	38	2188	173	2412
Gains: By election			168	17	185
change of status			12		12
Losses: By resignation			36	7	43
dropping			93	6	99
change of status				12	12
death	1		36	1	38
Total gains			180	17	197
Total losses	1		165	26	192
Present membership	12	38	2203	164	2417

The list of deaths comprises the names of A. Daubrée, honorary member, and the following members and associates: Matthew Addy (1884), R. B. Allen (1894), Maurice Chaper (1879), C. Jefferson Clark (1890), H. E. Collins (1876), Walter Crafts (1871), W. P. Darling (1891), R. T. Dodson (1892), G. W. G. Ferris, Jr. (1887), G. E. Fulton (1891), W. H. Furlonge (1888), W. C. Hadley (1883), Wm. Hainsworth (1888), Hakon Hammer (1889), Thos. Hodgson (1885), J. F. Holloway (1875), G. F. Hosking (1891), Thomas Howard (1886), Wm. E. Loane (1891), C. W. Marsh (1887), Chas. H. Miller (1892), I. R. Moister (1891), Norman J. Nichols (1893), Wm. E. Patterson (1888), J. W. Plummer (1880), N. W. Pratt (1886), Alfred Rickard (1884), Reuben Rickard (1884), Thomas Seddon (1888), C. O. Shields (1888), J. M. Silliman (1871), C. A. Stetefeldt (1881), A. Trippel (1883), Herman Veeder (1880), Joseph D. Weeks (1875), Thomas D. Whitaker (1892), and John A. Wilson (1876).

Of the above list, the names of Daubrée, Weeks, Stetefeldt, Holloway and Trippel have been, or will be, appropriately celebrated in special Biographical Notices in the *Transactions*.

Justus M. Silliman, for twenty-five years Professor of Mining Engineering and Graphics at Lafayette College, was one of the earliest and most highly esteemed members of the Institute, which he joined in 1871, the first year of its existence. He had contributed to the *Transactions* the following valuable papers: "Instruments for Projection-Drawing" (*Trans.*, x., 261); "The Kai-Ping Coal-Mine, North China" (*Trans.*, x., xvi., 95); "A Water-Manometer and Anemometer" (*Trans.*, xvii., 66).

Another veteran member was Mr. Walter Crafts (1871), with whose admirable personal character and professional ability in his special field of the charcoal-iron manufacture many of us were familiar.

Mr. W. H. Furlonge, though, residing in another hemisphere, he never attended a meeting of the Institute, will be remembered as the author of an interesting paper, "Notes on the Geology of the DeKaap Gold-Fields in the Transvaal" (*Trans.*, xviii., 334).

Messrs. Alfred and Reuben Rickard and J. W. Plummer were mine-managers of acknowledged eminence.

The Council has received the following communication:

SOUTH BETHLEHEM, PA.

At the meeting of the Association of American Steel Manufacturers held in New York on October 23, 1896, the following resolutions were passed:

1. *Resolved*, That we, the Association of American Steel Manufacturers, endorse the Decimal System as the proper standard for measuring all materials.

2. *Resolved*, That the Secretary be requested to forward a complete copy of the Committee's report, together with a copy of these resolutions, to the Secretaries of the American Institute of Mining Engineers, the American Society of Civil Engineers, the American Society of Mechanical Engineers, and the American Railway Master Mechanics' Association, as an evidence of the appreciation of the work accomplished by these societies towards the establishment of the Decimal System of Gauging, and as a proof of the hearty co-operation of this Association in this movement.

In the absence of the Secretary, the writer was made Secretary *pro tem.*, and encloses a copy of the Committee's report on Gauges, in accordance with the above resolutions. As an evidence that this endorsement of the Decimal System of Gauging carries considerable weight, the following list of members of the Association is quoted:

The Bethlehem Iron Co.	Otis Steel Co., Ltd.
Cambria Iron Co.	Pacific Rolling Mill Co.
Carbon Steel Co.	Paxton Rolling Mills.
The Carnegie Steel Co., Ltd.	Park Bros. & Co.
Catasauqua Mfg. Co.	Passaic Rolling Mill Co.
Central Iron Works.	Pennsylvania Steel Co.
Cleveland Rolling Mill Co.	Pottstown Iron Co.
Colorado Fuel & Iron Co.	Pottsville Iron & Steel Co.
Glasgow Iron Co.	Reading Rolling Mill Co.
Illinois Steel Co.	Schoenberger Steel Co.
Jones & Laughlins, Ltd.	Spang Steel & Iron Co.
Lukens Iron & Steel Co.	Worth Brothers.

Please acknowledge the receipt of this communication, and also bring the resolutions passed by our Association to the attention of the members of your Society at their next regular meeting.

Yours truly,

ALBERT LADD COLBY,
Secretary pro tem. A. A. S. M.

The report to which reference is made in the above communication will be presented to the Institute in substance, in a paper prepared, at the request of the Council, by the Secretary. Under Rule VII., as lately amended, neither the Council nor the Institute can take formal action for or against any proposed improvement of this kind; but it is deemed both permissible and advisable to lay before the Institute, for the information of its members, proposals, arguments and facts of professional importance.

President E. G. Spilsbury then delivered the Presidential Address on Recent Progress in Mining and Metallurgy.

After the adoption of the usual resolution instructing the Secretary to make suitable acknowledgment of courtesies received, the meeting was adjourned.

Members and Associates Elected.

The following persons were made members or associates by election during the sessions of the meeting:

Members.

Henry Bruen Alexander,	Sandon, British Columbia.
Edward P. Arthur, Jr.,	Cripple Creek, Colo.
George Percy Ashmore,	Caylloma, Peru.
W. M. Ayers,	South Hanover, Mass.
C. E. Bailey,	Virginia, Minn.
Frederick P. Bayles,	Sopris, Colo.
William Beals, Jr.,	Boston, Mass.
James S. de Benneville,	Philadelphia, Pa.
Frank A. Blakeslee,	De Lamar, Nev.
Collins P. Bliss,	New York City.
Rufus Buck,	Rossland, British Columbia.
Frederick Peck Burrall,	Custer City, So. Dakota.
Joseph Campbell,	London, England.
Robert Chapman,	Washington, D. C.
Thomas K. Code,	San Francisco, Cal.
Luis G. Cuevas,	San Luis Potosi, Mexico.
Thomas N. Davey,	Carthage, Mo.
George Lawrence Davison,	South Chicago, Ill.
John C. Dickey,	Central City, Colo.
James Arthur Diggles,	Angels Camp, Cal.
James M. Dikeman,	Berkeley, Cal.
William B. A. Dingwall,	Matehuala, Mexico.
Albert Doerr,	Freiberg, Germany.
Patrick J. Donohue,	Robinson, Utah.
T. C. DuPont,	Johnstown, Pa.
Thomas Ingleby Dyson,	Rockhampton, Queensland, Aust.
John W. Gates,	Chicago, Ill.
Frank Haas,	Columbus, O.
R. John Harvey,	Johannesburg, So. African Rep.
Edwin Chapin Holden,	New York City.
Louis D. Huntoon,	Paterson, N. J.
Charles Nicholas Jenks,	Sapphire, N. C.
Edward Mackay Johnson,	Argentine, Kas.
Thomas H. Kane,	Silverton, Colo.
Edwin Boyce Kimball,	Haywards, Cal.
Edward C. Koch,	New York City.

Louis John Krom, . . . Waterbury, Conn.
Frank M. Kurie, . . . Victor, Colo.
Wilbur L. Libbey, . . . North Brookfield, Nova Scotia.
Frank A. Lilliendahl, . . Zacatecas, Mexico.
Arthur Llewellyn, . . . Oorgaum, South India.
De Witt Loomis, . . . Detroit, Mich.
R. J. McCartney, . . . Silverton, Colo.
Edward J. Mathews, . . . Kaslo, British Columbia.
John Harrison Means, . . San Francisco, Cal.
John Miller, ... Julian, Mexico.
Dean Prescott Mitchell, . . Palo Alto, Cal.
Charles T. Mixer, . . . Ishpeming, Mich.
George Moore, ... Argentine, Kas.
Jacob H. Neff, ... Colfax, Cal.
George B. Nichols, . . . Chicago, Ill.
Edwin Norton, ... Chicago, Ill.
William Orr, ... Denver, Colo.
R. Oshima, ... Sapporo, Japan.
Fred. Wilson Paul, . . . Glasgow, Scotland.
Herbert Gerald Payne, . . West Norwood, England.
George W. Pfeiffer, . . . Philadelphia, Pa.
A. H. Purdue, ... Fayetteville, Ark.
Benjamin Risley Putnam, . Victoria British Columbia.
George Anderson Richard, . Rockhampton, Queensland, Aust.
Harry Roberts, ... Duluth, Minn.
Charles Edward Rogers, . . London, England.
J. W. Sauer, ... Salina, Colo.
Ferdinand A. Schiertz, . . Sinaloa, Mexico.
Richard C. Shaw, . . . Isabella, Cal.
Henry B. Shields, . . . Girard, Ohio.
Herbert S. Stark, . . . Johannesburg, So. African Rep.
James M. Stocker, . . . Atlanta, Ga.
Bradley Stoughton, . . . Boston, Mass.
William Effingham Sutton, . Monte Cristo, Wash.
Knox Taylor, ... Ketchum, Idaho.
William Willard Taylor, . . Iron Gate, Va.
J. Harry Watt, . . . Rankin Station, Pa.
William Lord Watts, . . San Francisco, Cal.
Charles A. Weck, . . . Douglas Island, Alaska.
Charles D. Wilkinson, . . Gaylord, Mont.
A. D. Wilson, ... Deadwood, South Dakota.

ASSOCIATES.

Fred. L. Champlain, . . Chicago, Ill.
Samuel L. Chisholm, . . Chicago, Ill.
Myron J. Church, . . . Golden, Colo.
B. I. Drake, ... South Bethlehem, Pa.
Alexander Forsyth, . . . Cambridge, Mass.
Rowland F. Hill, Jr., . . New York City.
William Carr Howard, . . Denver, Colo.

Fred Jost,	San Francisco, Cal.
Julius Kahn,	New York City.
Isaac William Kingsbury,	Boston, Mass.
D. F. Morgan,	Minneapolis, Minn.
Edward E. Noon,	Oro Blanco, Ariz.
Harry Thrie Poindexter,	Dillon, Mont.
Harold Rickard,	London, England.
John Smith Ross,	Lead, So. Dakota.
Alfred Douglas Sanders,	London, England.
Walter H. Summers,	Stone Mountain, Ga.
John Lynne Wharton,	Melbourne, Australia.
Bruce Clarke White,	Chicago, Ill.
G. C. Winslow, Jr.,	Boston, Mass.

Associates Made Members.

William D. Gordon,	Johannesburg, So. African Rep.
Rea E. Maynard,	Los Angeles, Cal.

Excursions and Entertainments.

On Wednesday evening, February 17th, a delightful reception was given to visiting members and guests of the Institute by the Technical Club at its handsome rooms, 228 Clark Street.

On Thursday afternoon a special train on the Illinois Central Railroad conveyed a large party to the Field Columbian Museum, which occupies the former Art Building of the Columbian Exposition in Jackson Park. This institution, of which Mr. F. J. v. Skiff, a member of the Institute, and formerly Director of the Department of Mines and Mining in the Columbian Exposition, is the Director, surprised and charmed every visitor by the magnificence, extent and interest of its varied collections.

On Thursday evening a subscription dinner was enjoyed in the Auditorium Hotel. On the same evening a session of the Western Society of Engineers was held at the Armour Institute, at which a paper was read by Mr. Ralph Modjeski on The Rock Island Bridge, and to which members of the Institute were invited.

Friday morning a special train on the Illinois Central Railroad conveyed a party comprising many Chicago friends, as well as the visiting members and guests of the Institute, to the shops of the said railroad at Burnside. After inspecting these

complete and admirable shops the party was transported to the works of the Illinois Steel Company at South Chicago, where, after luncheon, the extensive and interesting plant of the company was inspected.

Optional excursions were made upon special invitation to the following works and institutions:

The Art Institute, the Chicago Academy of Sciences, the University of Chicago, the Armour Institute, Fraser & Chalmers, the Gates Iron Works, the Crane Elevator Co., Chisholm, Boyd & White, and the Western Electric Co.

It is worthy of note that, in addition to the successful results of the excellent arrangements of the Local Committee, their happy choice of hotel headquarters, and the cordial co-operation of the Illinois Central Railroad Company, the Illinois Steel Company, and other institutions and corporations, the meeting and its excursions were favored with the best of weather.

Members, Associates and Guests Registered.

The following persons were registered at hotel headquarters:

James W. Abbott.
R. Wilson Anderson.
D. H. Bacon.
Lewis K. Bain.
H. P. Bellinger.
Charles E. Billin.
E. F. Brown.
R. H. Bulley.
M. C. Bullock.
E. D. Campbell.
J. A. Carney.
T. V. Church.
F. W. Clark.
C. W. Claybourne.
W. B. Cogswell.
F. K. Copeland.
W. M. Courtis.
Benedict Crowell.
G. M. Davidson.
F. W. Denton.
L. E. Dunham.
A. D. Edwards.
F. A. Emmerton.
Austin Farrell.
A. W. Fiero.
William Forsyth.
E. L. Foucar.
Fred W. Gordon.
G. A. Grube.
Ossian Guthrie.
Charles S. Hall.
D. G. Hearne.
Eric Hedburg.
Hiero B. Herr.
Henry D. Hibbard.
H. L. Hollis.
W. A. Holt.
William Hoskins.
Nelson P. Hulst.
Robert W. Hunt.
W. H. Jacques.
Guy R. Johnson.
James E Jopling.
William Kelley.
P. Larsson.
H W. Lash.
A. R. Ledoux.
James F. Lewis.
N. W. Lord.
R. S. MacPherron.
F. W. Matthiessen.
George Merryweather.
John W. Meier.
P. N. Moore.

George P. Nichols.
William S. Noyes.
E. W. Parker.
John C. Parkes.
G. S. Patterson.
William H. Peirce.
Fred. Pelouze.
Edward W. Penfield.
Jonas J. Pierce.
R. W. Raymond.
George S. Rice.
Robert H. Richards.
T. A. Rickard.
J. D. Robertson.
Axel Sahlin.
George S. Scott.
F. J. v. Skiff.
Alfred Smedley.
F. C. Smith.
H. E. Smith.
William Sooy Smith.
E. Gybbon Spilsbury.
C. E. Stafford.
Bertrand S. Summers.
Knox Taylor.
W. J. Taylor.
Arthur Thacher.
William H. Van Arsdale
Leonard Waldo.
Charles H. Wellman.
S. T. Wellman.
Albert H. Wolf.

Proceedings of the Seventy-Third Meeting, Lake Superior, July, 1897.

Committees at Houghton, Mich.

General: William E. Parnall, *Chairman;* Brainerd T. Judkins, *Secretary.*

Executive: F. McM. Stanton, *Chairman;* Fred Smith, J. H. Rice, C. A. Wright, Z. W. Wright.

Transportation: R. G. Collins, *Chairman;* H. D. Haddock, John C. Shields, Joseph Croze, Joseph Hennes.

Entertainment: Hon. Thomas B. Dunstan, *Chairman;* F. McM. Stanton, Graham Pope, A. D. Edwards, Hon. Wm. Harry, Wm. Veale, James R. Dee, W. D. Calverly, B. T. Judkins, R. M. Edwards, A. F. Leopold.

Reception: Hon. T. B. Dunstan, *Chairman;* Hon. J. A. Hubbell, T. L. Chadbourne, F. G. Coggin, Jr., Jacob Baer, E. D. Petit, Edward Ryan, J. H. Hicok, Simon Beahan, M. E. Wadsworth, Fred Smith, Ernst Bollmann, S. B. Harris, R. H. Shields, A. I. Jones, M. Messner, J. S. Dymock, Allen F. Rees, J. L. Harris, Z. W. Wright, Joseph Grose, J. R. Dee, W. D. Calverly, H. J. Vivian, E. F. Sutton, J. H. Wilson, L. L. Hubbard, B. T. Judkins, F. W. Sperr, F. W. McNair, Graham Pope, J. R. Cooper, James Moore, W. A. Dunn, A. F. Leopold, W. E. Parnall, A. C. Lane, C. A. Marr, C. A. Wright, J. H. Rice, John C. Shields, Hon. Wm. Harry, A. E. Seaman, A. J. Scott, T. J. Prince, Joseph Bosch, R. B. Goodell, R. G. Collins, Edgar Kidwell, C. F. Moor, J. D. Cuddihy, R. M. Edwards, F. McM. Stanton, O. W. Robinson, James H. Seager, Fred Mackenzie F. W. Denton, T. D. Ryan, Dr. J. E. Scallon, H. D. Haddock, Joseph Hennes, A. D. Edwards, Wm. Veale.

Committees at Duluth, Minn.

Institute Local: George D. Swift, *Chairman;* D. H. Bacon, W. G. La Rue, W. A. McGonagle, John Mallman, J. U. Sebenius, A. J. Trimble, G. L. Woodworth, J. D. Gilchrist, W. J. Olcott, E. J. Longyear, W. R. Appleby, R. M. Bennett, C. M. Case, A. L. Crocker, F. W. Denton, F. F. Sharpless, Edwin Ball, H. V. Winchell, P. H. Conradson, J. D. Estabrook, W. K. Gibson, Louis Stockett, R. B. Green, C. E. Bailey.

Transportation: J. L. Greatsinger, *Chairman;* Thomas Owens, J. T. McBride, D. M. Philbin, J. W. Kreitter.

Citizens' Local: Captain Alexander McDougall, *Chairman;* Joseph Sellwood, G. A. Leland, O. D. Kinney, Charles d'Autremont, David T. Adams, M. H. Alworth, G. H. Chesebrough, A. D. Thomson, O. H. Simonds, T. W. Hugo, James Sheridan, B. J. Van Vleck, A. M. Miller, A. E. Humphreys, G. G. Hartley, H. M. Peyton, A. L. Ordean, Louis Rouchleau, James T. Hale, Frank Hibbing, E. R. Brace, B. F. Myers.

Reception Committee for the Iron Ranges.

W. C. Agnew, *Chairman;* Robert Angst, John Pengilly, George Wallace, A. J. Carlin, R. H. Channing, M. W. Burt, Mowry Bates, John Lawson, J. H.

Hearding, P. Mitchell, C. M. Boss, H. B. Sturtevant, M. McCarty, Wm. H. Knight, E. B. Hawkins, J. H. Pearce, Richard Webb, C. H. Martz, Jacob Shilling, John A. Redfern, Harry Roberts, W. E. Dickinson, G. A. St. Clair, Richard Sellwood.

Dr. R. W. Raymond, Secretary of the Institute, being prevented by illness from attending this meeting, Mr. Theodore Dwight was appointed Acting Secretary by the Council.

The opening session was held at the Armory Theatre, Houghton, Mich., on Monday evening, July 12th. In the absence of Mr. William E. Parnall, Chairman of the General Committee, a cordial welcome was extended to the Institute by Dr. M. E. Wadsworth to which Vice-President H. S. Chamberlain, presiding in the absence of the President, responded.

The following papers were read by the authors:

The Origin and Mode of Occurrence of the Lake Superior Copper Deposits, by Dr. M. E. Wadsworth, Houghton, Mich.

The Electrolytic Assay as Applied to Refined Copper, by George L. Heath, South Lake Linden, Mich.

The following paper was read by the author, and illustrated with lantern-views:

The Efficiency of Built-up Wooden Beams, by Prof. Edgar Kidwell, Houghton, Mich.

The following papers were presented in print by the Acting Secretary:

The Potsdam Gold-Ores of the Black Hills of South Dakota, by Frank Clemes Smith, Rapid City, S. Dak.

The Influence of Lead on Rolled and Drawn Brass, by Erwin S. Sperry, Bridgeport, Conn.

The Fullers' Earth of South Dakota, by Heinrich Ries, New York City.

The Clays and Clay-Working Industry of Colorado, by Heinrich Ries, New York City.

Notes on the Tin-Deposits of Mexico, by Walter Renton Ingalls, New York City.

The following papers were read by title:

Biographical Notice of Peter Ritter von Tunner, by R. W. Raymond, New York City.

The Michigan College of Mines, by Dr. M. E. Wadsworth, Houghton, Mich.

Some Statistics of Engineering Education, by Dr. M. E. Wadsworth, Houghton, Mich.

A Mine-Dam, by William Kelly, Vulcan, Mich.

Some Dike-Features of the Gogebic Iron-Range, by C. M. Boss, Bessemer, Mich.

Notes on Six Months' Working of Dover Furnace, Canal Dover, Ohio, by Arnold K. Reese, Sparrow's Point, Md.

Investigations of Water-Supply, by F. H. Newell, Washington, D. C.

The Iron-Ore Supply, by John Birkinbine, Philadelphia, Pa.

The Marquette Iron-Range.— Its Discovery, Development and Resources, by J. E. Jopling, Marquette, Mich.

A Combination Retort and Reverberatory Furnace, by Courtenay DeKalb, New York City.

The Technology of Cement-Plaster, by Paul Wilkinson, St. Joseph, Mo.

The session was then adjourned.

The second and final session was held at Tower, Minn., on Wednesday evening, July 14th, Vice-President Chamberlain in the chair. After an address of welcome by the Hon. Joseph B. Colton, responded to by Mr. Oliver Williams, the following papers were read and discussed:

The Development of Lake Superior Iron-Ores, by D. H. Bacon, Soudan, Minn.

Mining Methods on the Mesabi Range, by C. E. Bailey, Virginia, Minn.

Explorations on the Mesabi Range, by E. J. Longyear, Hibbing, Minn.

The session was then adjourned.

Excursions and Entertainments.

For the Eastern members, who joined the steamer-trip on the Great Lakes, organized by Dr. David T. Day, the sight-seeing commenced on the 8th of July, when a considerable party took the "Black Diamond Express" on the Lehigh Valley Railroad from New York, others joining it at South Bethlehem and points in the Lehigh Valley.

On the morning of the 9th a chartered trolley-car carried about fifty of the party to the Carborundum Works, above Niagara Falls, where the manufacture of this new abrasive ma-

terial was inspected with great interest. It is obtained from a mixture of sand, sawdust, salt and coke by maintaining an electric current of a thousand horse-power through a 6-by-6-foot bank of this material, 16 feet long. Carborundum exceeds sapphire in hardness.

From here the party proceeded to the power-house of the Niagara Power Company, where the four 5000 H. P. electric generators were seen, and an inspection was made of the excavation intended for the additional turbines which will run six generators, of like capacity, that are now being added to the power-plant, already capable of developing 20,000 H. P.

From the power-house the car was taken to the International Hotel, where a luncheon was served.

During the afternoon a trolley-trip was made on the Canadian shore above the Falls and then down past the rapids to Lewiston, where the river was crossed. Another special trolley-car carried the party up the magnificent Gorge Road on the American side.

For the arrangement of this interesting and enjoyable excursion, the party is indebted equally to Dr. Day and to Mr. C. M. Hall, of the Pittsburgh Reduction Co.

Returning to Buffalo on the special car, the steamer North-West was boarded for the trip through the Great Lakes to Houghton.

At Cleveland, the next morning, the party was met by a committee of the local members, headed by Mr. S. T. Wellman, with tally-ho coaches, and a ride through the beautiful residence-streets, Euclid Avenue and Prospect Street, was much enjoyed.

The original party was considerably augmented at Cleveland, Detroit and Mackinac, at which latter place carriages were secured for a ride around the interesting island.

Arriving outside the harbor of Houghton shortly after nine o'clock on the morning of July 12th, the steamer was met by the Local Committee with two tugs; and to insure a safe transfer, the life-saving crew was on hand. Proceeding to the dock, at the head of the Portage Canal, the party was met by a brass band composed of miners from the Quincy mine, and by a large number of the Local Committee. Here a transfer was made to a large double-deck barge, which was promptly taken in tow, and the following itinerary was carried out:

10.30 A.M. Arrive at Lake Superior Smelting Works.
11.30 A.M. Leave Lake Superior Smelting Works. Lunch on Barge.
12.00 M. Arrive at Tamarack Osceola Copper Manufacturing Co.'s Works, Dollar Bay.
1.00 P.M. Leave Dollar Bay.
1.20 P.M. Arrive at Tamarack Stamp Mills.
2.20 P.M. Leave Tamarack Stamp Mills.
3.00 P.M. Arrive at Tamarack Mine.
5.30 P.M. Leave for Houghton.

Special mention should be made of the luncheon which was served on the barge by the ladies of Houghton while *en route.*

Tuesday morning, July 13th, the party divided into two sections, one to visit the Quincy and Franklin mines, under the guidance of Capt. S. B. Harris, the manager of the Quincy property. After seeing the various power-plants and mills of the two companies, about 20 went down the deep shaft of the Quincy, some 3800 feet below the surface. The other division visited the Atlantic mines, where, after an inspection of the property, a luncheon was enjoyed through the hospitality of Mr. F. McM. Stanton. The early afternoon was spent in examining the thorough equipment of the Michigan College of Mines. At 4 P.M. the barge met the party at the Isle Royale dock, near the college buildings, proceeding to the Onegaming Club House, some miles distant on the beautiful shore of Lake Linden. There an elaborate supper was served to the visiting guests, waited upon by the members of the Club. After the repast, Capt. H. S. Chamberlain, as presiding officer, called for a number of toasts. A graceful address of welcome was delivered by the Hon. T. B. Dunstan, Lieut.-Governor of Michigan and Chairman of the Houghton Entertainment Committee, followed by Col. Graham Pope, Judge Jay A. Hubbell, Hon. Porter King, Dr. M. E. Wadsworth, Prof. Ira Remsen, Charles Kirchhoff and Dr. David T. Day, the speakers for the visiting members endeavoring to express the warm appreciation and gratitude of the party for the remarkable hospitality and the many courtesies extended to them during their stay in the copper district.

At 9 P.M. the special train of Wagner sleeping-cars was taken for Duluth. Through the kindness of Messrs. C. E. Bailey and E. J. Longyear, of the Duluth Local Committee, who had joined the party upon its arrival at Houghton, the arrangements for the sleeping-car accommodations had been made in advance, so that all were able to enjoy a much-needed rest on the train. On arriving at Duluth at 9 o'clock Wednesday morning, the party was met by many members of the Local Committee. Here dining-cars were attached, and in a short time the train proceeded on a special schedule, over the Duluth and Iron Range Railroad, to Two Harbors, where the ore-docks were inspected. From there the train carried the party to the mines of the Minnesota Iron Co. at Soudan, on the shore of Lake Vermilion, where this interesting plant was seen, many of the party going underground. Owing to the lateness of the hour, the projected visit to the mines at Ely was abandoned, and the session scheduled to occur there was transferred to Tower, where the train was held for the night.

The business meeting was opened by an eloquent address of welcome by Hon. J. B. Cotton, of Duluth, responded to on behalf of the Institute by Mr. Oliver Williams.

The following morning, Thursday, July 15th, the train proceeded to the Mesabi range, where the first stop was made to examine the Biwabik and the adjoining Duluth, Canton and Cincinnati mines. From this interesting group the journey was made to the Fayal mine of the Minnesota Iron Co. Here a number of the party went underground after inspecting the surface-plant, and from thence proceeded to the Adams mines of the Consolidated Co. At the Auburn mine a short stop was made to see this interesting example of the "milling" system of mining.

On reaching the Franklin mine, some miles beyond, the train was for a time abandoned to be met again at Virginia. Between these points the Norman, Commodore, Ohio, Lone Jack and Oliver mines were passed. These mines are all employing the stripping-method in its various modifications.

Some of the party took the train again, reaching Mountain Iron by way of Wolf, while others enjoyed a bicycle ride across country through the kindness of Mr. C. E. Bailey, of Virginia,

Minn., who provided the wheels. Both parties arriving at the Mountain Iron mine together, a brief inspection was made of this great property, from which a million and a quarter tons of ore have been taken. Boarding the train again, a quick trip was made back to Duluth.

As examples of the various types of Minnesota mines had been examined, together with the different methods employed in the ranges in mining the ore, and as the Hibbing district had so recently suffered from the heavy rains, this part of the trip was abandoned. The mines there were badly flooded and it would have been impossible to see them favorably.

The party was indebted for the special trains through the Iron Ranges to the Duluth and Iron Range Railroad, and to the Duluth, Missabe and Northern Railroad.

The Duluth, Mississippi River and Northern Railroad had extended the courtesies of their road, but as the trip to Hibbing did not take place, the privilege was not availed of. The Duluth Street Railway Companies courteously provided each of the visiting members with a series of tickets good on their lines during the meeting.

On July 16th a large number of carriages were in waiting to take the party on the Boulevard drive up the mountain side, where a magnificent view of Duluth harbor and the lake was had, and on the return the beautiful gorge scenery was enjoyed. A Lake trip was taken by other members of the party during this time.

Many establishments in and near Duluth, including the American Steel Barge Works, the Imperial Mill, Elevators and Coal Docks, Duluth Superior Bridge, the Board of Trade, the Duluth High School, and the Mitchell & McClure and Merrill & Ring Saw Mills, courteously extended a welcome to such members of the Institute as should desire to inspect their plants. Friday evening, July 16th, a reception was given by the Local Committee at the Spalding Hotel.

Saturday, July 17th, at 10.45 A.M., a party of thirty left for the Black Hills, South Dakota, and at 1.45 P.M. a larger number took the steamer North Land for the East.

The Local Committee provided an itinerary with so much useful and valuable information relating to the iron industry,

that it has been thought advisable to append extracts from it to these *Proceedings* for the benefit of the members at large.

The maps with which it was provided are not reproduced, as Prof. Denton's paper on Methods of Iron Mining in Northern Minnesota supplies this want.

Through Mr. N. H. Winchell, State Geologist, copies of the *State Geological Survey Bulletin*, No. 10, were supplied to all the guests; and copies of the *News Tribune Annual* were furnished through the kindness of Mr. A. T. Thoits.

Visit to Black Hills, South Dakota.

The trip to the Black Hills of South Dakota, being largely in the nature of an excursion, the description of it will be treated in a correspondingly informal manner. Prof. F. C. Smith had promised the members that a hearty welcome would be extended to those who should visit this interesting district after the close of the Lake Superior meeting, and, not satisfied with awaiting their arrival, Dr. V. T. McGillycuddy, Mayor of Rapid City and President of the School of Mines there, journeyed the 1200 miles to Duluth to greet the members and escort them back. The Acting Secretary undertook to "personally conduct" the party, but his duties were made light by the combined co-operation of the officials of the North Western lines and of Dr. McGillycuddy.

During the long run over the splendid roads of the North Western system, not a single mishap and not a moment's delay occurred. These Railroad Companies not only gave to the members reduced rates, but made them their guests in the Black Hills and hauled the Wagner car, Paoli, free through many miles of beautiful country.

Dr. McGillycuddy was tireless in planning for the pleasure and comfort of his guests, and those who took part in the trip will not readily forget his kindness. The following pleasant company filled the sixteen sections of the Paoli, and stayed together, with few exceptions, until Buffalo was reached on the return.

Mr. and Mrs. I. P. Pardee,
Mr. and Mrs. W. S. Ayres,
Mr. and Mrs. T. D. Jones,
Mr. and Mrs. D. W. Brunton,
C. W. Goodale,
L. Holbrook,
J. B. Randol,
The Misses Randol,
Z. T. Sowers,
William Sowers,
Miss Sowers,
V. T. McGillycuddy,
W. S. De Camp,
Leo De Camp,
Miss De Camp,
Porter King,
Ira Remsen,
J. Rand, Jr.,
A. W. Sheafer,
S. H. Pitkin,
G. M. Davidson,
H. A. J. Wilkens,
Francis T. Freeland,
E. W. Parker,
Theodore Dwight.

The first stop of interest was made at Spooner, Wis., where a delicious luncheon was enjoyed, after which the train was boarded for St. Paul. Excellent meals were provided all along the route, when telegrams were sent ahead to announce the coming of the Mining Engineers.

After a three-hour stop at St. Paul and a night on the train, Omaha was reached, and the six hours there were utilized in visiting the Omaha and Grant Smelter. Arriving the next morning, July 19th, at Rapid City, the train was met by a number of citizens, who during the next twenty-four hours, with the active assistance of the ladies, entertained the party with true western hospitality. Carriages were provided for a visit to the School of Mines, returning in time for a hearty luncheon served by the ladies. A picnic to the beautiful Black Cañon was enjoyed during the afternoon. Leaving hospitable Rapid City the next morning, Deadwood was reached a few hours later. Here again many citizens, with Prof. F. C. Smith at their head, met the visitors and escorted them to the Bullock House. After luncheon an interesting and beautiful trip was made on a special train, furnished through the courtesy of the railroad, over the narrow gauge, winding up into the mountains, and passing innumerable mine-tunnels and shafts. Stops were made to enjoy particularly beautiful bits of scenery, and, on the return, to explore the tunnel at the Stewart mine of the Golden Reward Co. As this was the only mine which the party were enabled to enter during their stay in the Black Hills, it was examined with much interest. The following morning was spent in seeing the great mills and surface-plant of the Homestake Mining Co. at Lead. In the afternoon Prof. Smith showed the party through the chlorination-works

of the Golden Reward Co. at Deadwood, of which he is general manager.

The third and last day, the Mining Engineers were the guests of the Burlington and Missouri River Railroad Co., which hauled two special cars full of the members and their hosts (many of whom took part in this trip) through the beautiful winding cañon of Spearfish creek. There are few more glorious railroad-rides in the world, and none that can be enjoyed with greater comfort. The 35 miles of broad-gauge track, mounting to an altitude of 7000 feet, is an impressive example of engineering skill. It was built at a cost of $1,400,000. The views afforded of the beautiful Black Hills country from many points on the journey are both magnificent and inspiring. After luncheon at Spearfish, the return trip was made. Leaving Deadwood that evening at 6 o'clock, Hot Springs was reached by 10.30, when all hands adjourned to the famous Hot Springs plunge, which had been kept open specially, and refreshed themselves after the journey. The next day was spent by the majority in resting, after the long round of sight-seeing and entertainment. On Friday, July 23d, the last day in the Black Hills country, a ride of 12 miles was made by coach and horseback to the wonderful Wind Cave, with its 80 or 90 miles of passages running in many parallel rows and tiers, connected by gallery and stope, like the underground system of a great coal-mine. The peculiar form of the deposits on the roof and walls lacks the familiar cylindrical stalactite and stalagmite, being what is locally called "lace-work." The richly varying colors in this cave add greatly to the beautiful outlines of its scenery, if the expression may be used. Returning again to Hot Springs, the train was taken some hours later for the East.

It was expected that the sight-seeing of the trip would end here, but upon reaching Minneapolis it was found that Messrs. Estabrook, Woodman and Winchell had prepared an elaborate itinerary for seeing the local points of interest, including a trip on a chartered car to Minnehaha Falls, and by rail to Lake Minnetonka, with a steamer-ride on this wonderfully beautiful body of water, with its 266 miles of shore. After a dinner at the West Hotel and an evening of music, through the kindness of Mr. and Mrs. Winchell, the party proceeded on its journey to Duluth to take the steamer North West for Buffalo and

home. The voyage was a repetition of the pleasant experience of three weeks before, and at its end the party, who had made warm friendships on their long journeyings, regretfully bade each other farewell.

Members and Associates Elected.

The following persons were elected as members or associates by mail-ballot, April and July, 1897:

In April.

Members.

Noble C. Banks,	Graham, N. M.
Edward Blum,	Buenaventura, Colombia, S. A.
William J. Burns,	Pittsburgh, Pa.
William Young Campbell,	Johannesburg, So. African Rep.
Felipe Gomez del Campo,	San Luis de la Paz, Guanajuato, Mexico.
Allan J. Clark,	New York City.
E. H. Dewey,	Dewey, Idaho.
John B. Fleming,	Robinson, Utah.
Ernst Harms,	El Paso, Tex.
Eugene Lawrence Messler,	Pittsburgh, Pa.
Horatio Throop Morley,	Detroit, Mich.
Thomas Neilson,	Keswick, Cal.
Edwin Wilder Newton,	Bolivar, Mo.
Charles Of,	New York City.
S. F. Parrish,	Leadville, Colo.
Godwin H. Powel,	Powellton, W. Va.
William Peile,	Johannesburg, So. African Rep.
R. Neil Smith,	Kalgoorlie, W. Australia.
George McLeod Spotswood,	Kingston, Ontario.
Alfred I. Totten,	Tallapoosa, Ga.
Godfrey T. Vivian,	Pachuca, Mexico.
Louis Webb,	Buluwayo, So. Africa.
J. H. Weddle,	Leadville, Colo.
Edward Gilbert Williams,	Colon, Colombia, S. A.
Alexander Hamilton Wood,	Nashville, Tenn.

Associate.

Porter King,	Atlanta, Ga.

Change of Status.

George F. C. Hosking,	Auckland, New Zealand.

In July.

Members.

J. Ahbe,	Tower, Minn.
Harry Kenneth Austin,	Johannesburg, So. African Rep.

James A. Beckett,	Hoosick Falls, N. Y.
C. W. Benton,	San Lucia, Honduras, C. A.
Rene de Bonaud,	Auckland. New Zealand.
Archibald Graham Brownlee,	Chicago, Ill.
Alvin B. Carpenter,	Chihuahua, Mexico.
Henry Edward Cranmer Carry,	Nelson, B. C. Canada.
George Plunkett Chaplin,	London, England.
George L. Cheesebrough,	Duluth, Minn.
J. H. Chewett,	Toronto, Canada.
John G. Cohoe,	Victor, Colo.
Spencer Cragoe,	London, England.
F. E. Dodge,	Flushing, L. I., N. Y.
Auguste Dutreux,	Luxembourg.
F. N. Esselstyn,	Ishpeming, Mich.
Walter Menzies Fulton,	Coromandel, New Zealand.
Frederick S. Harris,	Union de Tada, Mexico.
Harold Justice Harris,	London, England.
R. T. Judkins,	Houghton, Mich.
Edmund Juessen,	Elk City, Idaho.
Edgar Kidwell,	Houghton, Mich.
Hugh Mackenzie Leslie,	Oorgaum, India.
Sydney Harland Loram,	Tallal, Chili, S. A.
Charles Gordon Lyon,	Melbourne, Australia.
M. E. McCarthy,	Virginia, Minn.
James W. McCoy,	Baker City, Ore.
John Egbert McCurdy,	Chihuahua, Mexico.
Alexander John MacGeorge,	Kalgoorlie, W. Australia.
C. H. Martz,	Duluth, Minn.
Charles A. Matcham,	Phillipsburg, N. J.
John W. Mercer,	Cripple Creek, Colo.
Frederick Horace Minard,	Denver, Colo.
Pentecost Mitchell,	Hibbing, Minn.
Arthur Mosley,	Dunedin, New Zealand.
John C. Moulden,	Broken Hill, New South Wales.
Charles M. Myrick,	San Francisco, Cal.
Robert Kitching Painter,	San Juancito, Honduras, C. A.
James Park,	Thames, New Zealand.
J. L. Parker,	Rossland, British Columbia.
William A. Parker,	Chicago, Ill.
William Chapman Potter,	Chicago, Ill.
H. A. Pringle,	Rat Portage, Canada.
Ira Remsen,	Baltimore, Md.
Francis C. Robbins,	Leadville, Colo.
H. Röchling,	Völklingen, Germany.
William T. Shaw,	Gilt Edge, Mont.
Rollins C. Steese,	Youngstown, Ohio.
Augustus Wynne Stoddart,	Johannesburg, So. African Rep.
Albert Ernest Thomas,	Coolgardie, W. Australia.
Paul Wilkinson,	St. Joseph, Mo.
Roberto Yarto,	Guanajuato, Mexico.

Associates.

Harry J. Boyd,	Bel Air, Md.
John A. Rogers,	Neosho, Mo.
Augustus Simson,	Launceston, Tasmania.

Howard R. Stewart,	Zeehan, Tasmania.
Edgar A. Weimer,	Lebanon, Pa.

Change of Status.

W. S. Davidson,	New York City.
R. Henry Jeffrey,	London, England.
E. B. Kirby,	Denver, Colo.

Members, Associates and Guests Registered During the Meeting.

D. T. Adams,
W. C Agnew,
M. H. Alworth,
Robert Angst,
W. R. Appleby,
Charles d'Autrement,
W. S. Ayres,
D. H. Bacon,
Jacob Baer,
C. E. Bailey,
Edwin Ball,
Mowry Bates,
Simon Beahan,
R. M. Bennett,
E. Bollmann,
Joseph Bosch,
C. M. Boss,
E. R. Brace,
J. Bunnell,
M. W. Burt,
W. S. Brown,
D. W. Brunton,
W. D. Calverly,
A. I. Carlin,
C. M. Case,
T. L. Chadbourne,
H. S. Chamberlain,
R. H. Channing,
G. H. Chesebrough,
F. G. Coggin, Jr.,
R. G. Collins,
H. W. Conrad,
P. H. Conradson,
J. R. Cooper,
Eugene Coste,
J. B. Cotton,
A. L. Crocker,
Joseph Croze,
J. D. Cuddihy,
G. M. Davidson,
D. T. Day,
Leo De Camp,
W. S. De Camp,
J. R. Dee,
F. W. Denton,
J. M. Desloge,
W. E. Dickinson,
W. A. Dunn,
T. B. Dunstan,
T. Dwight,
J. S. Dymock,
A. D. Edwards,
R. M. Edwards,
J. D. Estabrook,
John L. Ferguson,
F. T. Freeland,
J. W. Fuller, Jr.,
W. K. Gibson,
J. D. Gilchrist,
C. W. Goodale,
R. R. Goodell,
J. L. Greatsinger,
R. B. Green,
G. D. Grubb,
H. D. Haddock,
J. T. Hale,
J. L. Harris,
S. B. Harris,
William Harry,
G. G. Hartley,
E. B. Hawkins,
J. H. Hearding,
Joseph Hennes,
Frank Hibbing,
J. H. Hicok,
Levi Holbrook,
L. L. Hubbard,
J. A. Hubbell,
T. W. Hugo,
A. E. Humphreys,
Axel Ihlseng,
A. I. Jones,
T. D. Jones,
Washington Jones,
B. T. Judkins,
W. J. Keep,

William Kelly,
Edgar Kidwell,
Porter King,
O. D. Kinney,
Charles Kirchhoff,
William H. Knight,
J. W. Kreitter,
A. C. Lane,
Per Larsson,
W. G. La Rue,
John Lawson,
J. H. Lee,
G. A. Leland,
A. F. Leopold,
John Lilly,
E. J. Longyear,
O. Lord,
C. H. Ludlow,
Frank Lyman,
J. T. McBride,
M. McCarty,
A. McDougall,
Fred. McKenzie,
W. A. McGonagle,
F. W. McNair,
G. M. Magie,
John Mallman,
G. A. Marr,
C. H. Martz,
F. W. Mathiessen,
M. Messner,
A. M. Miller,
P. Mitchell,
C. F. Moor,
James Moore,
B. F. Myers,
W. J. Olcott,
A. L. Ordean,
Thomas Owens,
I. P. Pardee,
E. W. Parker,
W. E. Parnall,
J. H. Pearce,
L. Peckitt,
John Pengilly,
E. D. Petit,
A. M. Peyborn,
D. M. Philbin,
S. H. Pitkin,
Graham Pope,
J. J. Prince,
J. R. Rand,
J. R. Rand, Jr.,
Theodore D. Rand,
J. B. Randol,
John A. Redfern,
A. F. Rees,
Ira Remsen,
J. H. Rice,
Harry Roberts,
T. W. Robinson,
Louis Rouchlean,
Edward Ryan,
T. D. Ryan,
J. H. Seager,
J. E. Scallon,
A. J. Scott,
G. A. St. Clair,
A. E. Seaman,
J. Uno Sebenius,
Joseph Sellwood,
R. Sellwood,
A. W. Sheafer,
F. F. Sharpless,
J. Sheridan,
J. N. Sherrerd,
R. H. Shields,
Jacob Shilling,
J. A. Shields,
O. H. Simonds,
C. D. Skillings,
Fred. Smith,
Z. T. Sowers,
William Sowers,
F. W. Sperr,
Erwin S. Sperry,
F. McM. Stanton,
Louis Stockett,
Bradley Stoughton,
H. B. Sturtevant,
E. F. Sutton,
George D. Swift,
A. D. Thomson,
A. J. Trimble,
B. J. Van Vleck,
William Veale,
H. I. Vivian,
M. E. Wadsworth,
George Wallace,
Richard Webb,
C. H. Wellman,
William Whitney,
H. V. Winchell,
J. F. Wilcox,
Paul Wilkinson,
Oliver Williams,
J. H. Wilson,
G. L. Woodworth,
C. A. Wright,
Z. W. Wright.

Reprinted from the Programme of the Meeting, Compiled by the Local Committee.

Shipments of Iron-Ore from the Vermilion and Mesabi Ranges in Gross Tons Down to 1897.

NAME OF MINE.	1884.	1885.	1886.	1887.	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	Totals.
Vermilion Range.														
*Chandler ..... U	.....	.....	.....	.....	54,612	306,220	336,002	373,969	651,655	435,930	558,050	605,024	471,545	3,793,007
*Minnesota ..... U	62,124	225,484	304,396	394,252	457,341	535,318	532,000	517,570	498,353	370,303	390,463	432,760	448,707	5,169,071
Pioneer ..... U	.....	.....	.....	.....	.....	3,144	12,012	3,079	2,651	.....	.....	40,054	149,073	210,013
Zenith ..... U	.....	.....	.....	.....	.....	.....	.....	.....	14,991	14,388	.....	.....	18,765	48,144
Total	62,124	225,484	304,396	394,252	511,953	844,682	880,014	894,618	1,167,650	820,621	948,513	1,077,838	1,088,090	9,220,235
Mesabi Range.														
†Adams ..... M	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	59,141	234,562	293,703
Ætna (Lowmore) ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	.....	1,645	.....	.....	17,723	19,368
*Auburn ..... M	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	108,210	376,970	131,478	616,658
Biwabik ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	.....	151,500	90,048	247,069	242,565	731,182
*Canton ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	24,416	213,853	359,020	16,261	613,550
Cincinnati ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	26,372	.....	17,187	57,324	100,883
Cloquet (Vega) ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5,628	47,700	96,280	149,608
Commodore ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	65,137	7,213	.....	22,063	94,413
†Duluth ..... M	.....	.....	.....	.....	.....	.....	.....	.....	.....	37,626	.....	.....	.....	37,626
*Fayal ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	136,601	248,645	385,246
Franklin Group ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	46,617	223,399	286,423	231,086	787,525
*Genoa ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	17,136	17,136
Hale ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,616	24,167	31,004	70,006	128,793
†Lake Superior ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	58,123	67,659	125,782
Mahoning ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	117,884	167,245	285,129
†Minnewas ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	13,858	2,140	.....	.....	15,998
**Mountain Iron (and Rath) ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	4,245	119,818	573,440	371,274	142,021	1,210,798
*Norman ..... M	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	38,999	93,392	77,523	209,914
†Ohio ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	28,943	69,925	98,868
**Oliver (I) ..... O. P.	.....	.....	.....	.....	.....	.....	.....	.....	.....	123,015	505,955	500,377	808,291	1,937,638
Sellers ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	47,433	153,037	200,470
Williams (N. Cincinnati) ..... U	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,046	11,249	14,295
Total	.....	.....	.....	.....	.....	.....	.....	.....	4,245	613,620	1,793,052	2,781,587	2,882,079	8,074,583
Miscellaneous	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,320
Grand Totals, both ranges	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,970,169	17,297,138

* Operated by the Minnesota Iron Co.
† Operated by the L. S. Consolidated Mines.
** Operated by the Oliver Mining Co.

I—Mesabi Mountain and Lone Jack.
U—Underground Mine.
M—Milling Mine.
O. P.—Open Pit Mine.

Shipments of Other Lake Superior Ranges.

	1896.	Total to 1897.
Marquette Range	2,604,221	46,538,187
Menominee Range	1,560,467	22,994,428
Gogebic Range	1,799,971	20,788,787
Grand Total for all other Ranges	5,964,659	90,321,402

Approximate Shipments by Ports in 1896.

From Two Harbors	1,714,406
From Duluth	2,258,355
From Ashland	1,573,334
From Marquette	1,565,172
From Escanaba	2,316,862
From Gladstone	216,363

Complete Analyses of Cargo Samples of Grades of Ore Mined in Minnesota.

Name of Mine.	Name of Grade.	Iron.	Silica.	Phos.	Mang	Sulp.	Alum	Lime.	Magn	Moist.
VERMILION RANGE.										
Minnesota	Minnesota	67.74	1.55	.048	None	None	.90	.27	.12	2.00
"	Soudan	65.31	4.36	.104	None	Trace	.68	.62	.28	.....
"	Red Lake	64.41	3.74	.108	None	Trace	1.81	.35	.11	.....
"	Vermilion	67.36	1.68	.156	None	None	.62	.60	.25	.....
Chandler	Chandler	64.70	4.26	.036	.13	Trace	1.37	.33	.10	5.79
"	Long Lake	60.59	7.80	.040	.13	.004	1.85	.72	.20	7.55
Pioneer	Pioneer	64.91	4.50	.035	.21	.020	1.60	.16	.07	8.11
"	Pilot	61.20	9.96	.030	.66	.006	1.62	.21	.10	8.50
Zenith	Zenith	66.02	2.95	.050	.14	.009	1.61	.12	.07	5.52
MESABI RANGE.										
Adams	Adams	64.18	2.80	.035	.40	.007	.80	.21	.10	7.29
Auburn	Auburn	65.04	2.25	.047	.39	Trace	1.47	.68	.15	9.56
"	Audrey	62.69	2.64	.071	.66	Trace	1.52	.25	.14	12.07
Biwabik	Biwabik	64.61	3.00	.033	.25	.005	.50	.24	.15	7.71
Burt	Burt	65.48	3.17	.034	.49	.006	1.43	.56	.20	8.97
Canton	Climax	63.64	1.94	.036	.87	Trace	.60	.88	Trace	8.65
"	Canton	60.80	4.25	.048	.49	Trace	.93	.71	.07	9.97
Cincinnati	Cincinnati	61.50	6.32	.037	.59	.015	.98	.34	.15	8.00
"	Cincin'ti Silica	59.00	8.00	.030	.60	.015	1.00	.34	.15	8.00
Cloquet	Cloquet	62.07	4.00	.033	.37	.....	1.60	.28	.30	13.13
"	Mangan	59.80	5.50	.034	.68	.....	1.52	.32	.10	13.55
"	Valley	61.08	5.66	.041	.31	.....	.....	.....	.....	12.76
Commodore	Commodore	63.60	4.10	.038	.12	.011	.90	.37	.04	10.25
Duluth	Berringer	61.16	4.55	.065	.35	.012	1.59	.11	.07	8.91
Fayal	Fayal	62.55	3.06	.037	1.00	Trace	1.30	.26	.14	9.55
Franklin	Franklin	62.79	4.12	.036	.69	.021	.81	.30	.26	6.26
Genoa	Genoa	62.70	3.75	.033	.65	None	1.16	.60	.19	11.29
Hale	Hale	60.00	5.00	.100	.55	.016	.52	1.58	.....	9.00
Lake Superior	Hull	65.52	2.43	.024	.19	.018	.79	.02	.14	10.79
"	Rust	65.00	2.50	.030	.25	.012	1.25	.25	.15	9.25
Mahoning	Mahoning	64.28	2.75	.050	.33	.020	1.50	.21	.14	10.34
"	Beaver	64.44	2.20	.081	.17	.012	1.50	.20	.10	10.61
"	Shenango	64.07	2.69	.060	.30	.020	1.62	.20	.16	10.53
Minnewas	Minnewas	65.63	2.35	.047	.35	.008	1.14	.20	.03	9.00
Mountain Iron	Mountain Iron	64.08	3.68	.043	.42	.010	1.80	.18	.10	12.27
"	Helmer	64.00	3.65	.051	.48	.015	1.80	.15	.10	11.15
"	Tubal	63.25	4.50	.065	.45	.013	1.75	.20	.08	11.54
Norman	Norman	62.15	3.75	.069	.90	Trace	1.82	.52	.13	8 41
"	Saxon	64.29	3.75	.039	.56	.....	1.21	.12	.04	8.00
Ohio	Ohio	61.53	4.50	.082	.40	.007	1.45	.38	.25	6.95
"	Ohio Katonah	64.41	2 95	.053	.40	.006	1.34	.37	.25	8.35
Oliver	Oliver	63.31	4.53	.053	.51	.010	2.10	.20	.047	9.07
"	Juniata	62.40	3.30	.085	.67	.013	4.62	.05	Trace	10.00
"	Preble	60.00	4.75	.080	1.67	.020	3.58	.10	.108	9.56
Sellers	Sellers	64.23	3.66	.036	.38	.019	.95	.19	.12	8.73
Williams	Williams	60.40	9.20	.036	.77	.011	.95	.42	.12	9.70

Complete Analyses of Cargo Samples of Ores from the Marquette Range, Michigan.

MARQUETTE RANGE.

Name of Grade of Ore.	Iron.	Silica.	Phos.	Mang.	Sulph.	Alum.	Lime.	Magn.	Moist.
Angeline, Hard	66.19	.....	.014	.....	.....	.....	.....	.....	8.99
" Hematite	66.08	.....	.042	.....	.....	.....	.....	.....	8.69
" South	62.87	4.45	.128	.27	.028	1.79	.88	.06	11.02
Barnum	65.50	3.49	.075	.36	.026	1.80	.32	.26	.89
Beacon	46.00	29.75	.043	.07	.030	2.50	1.00	.80	.84
Blue	63.40	5.20	.118	.26	.010	1.96	.55	.72	10.55
Buffalo	61.80	6.60	.127	.55	.008	1.84	.86	1.16	10.70
" South	61.95	5.70	.115	.39	.006	1.75	.92	.71	10.90
Cambria	61.40	9.60	.046	.34	.012	1.92	.74	.80	10.10
Cleveland Bessemer	67.28	2.75	.044	.11	.017	.20	.25	.24	.84
Cliff Shaft	62.42	3.65	.113	.35	.020	1.70	1.15	1.00	1.06
Columbia	59.13	8.20	.550	.06	.....	1.46	1.35	.06	10.77
East End Bessemer	61.06	7.48	.046	.33	.014	2.09	.23	.80	10.33
Essex	60.00	10.25	.110	.30	.020	2.20	.53	.20	.74
Foster	51.45	18.45	.142	.27	.020	1.10	.18	.39	3.63
" Silica	42.00	27.00	.122	3.86	.002	1.36	.20	3 67	3.00
Humboldt	65.90	3.91	.150	.13	.020	1.88	.88	.24	.....
Ishpeming	57.41	10.35	.072	.57	.030	2.62	.88	236	9.89
Jackson, South Side	48.97	30.81	.050	2.12	.....	2.00	.....	.....	.....
" Pit No. 7	57.77	11.00	.068	2.15	.....	2.00	.....	.....	.....
Lake	60.85	5.67	.088	.60	.018	2.10	.41	.32	12.10
Lake Bessemer, No. 1	65.44	.....	.030	.....	.....	.....	.....	.....	9.97
" No. 2	64.01	5.50	.038	.24	.019	1.40	.35	.21	9.63
Lake Silica	48.27	25.40	.042	.43	.035	1.69	.36	.47	9.07
Lake Superior No. 1	65.00	3.40	.100	.23	.018	2.10	.26	.24	1.15
Lillie	61.75	5.60	.077	.39	.010	1.92	.37	.08	9.60
Marquette	43.50	34.09	.051	.26	.021	.97	.17	.17	6.12
Michigamme	66.50	4.67	.130	.23	.050	.07	.25	.62	.75
Mwabe Friend	39.54	.....	.020	.....	.....	.....	.....	.....	.92
Negaunee	63.53	4.55	.057	.25	.025	2.64	.75	.15	10.67
" Non-Bessemer	62.30	4.30	.115	.30	.025	2.90	.92	.20	10.67
Non-Bess., Old Mine Hem.	62.00	5.00	.130	.45	.020	2.75	.53	.21	7.78
Norfolk	56.00	13.75	.050	.09	.030	2.50	1.10	1.05	1.12
Oxford	64.50	4.45	.075	.32	.020	1.80	.25	.23	1.14
Peninsula	59.00	9.20	.118	.31	.007	1.92	.82	.41	10.80
Platt	58.70	.....	.038	.....	.....	.....	.....	.....	8.78
Prince of Wales	61.55	6.10	.122	.36	.012	1.72	1 40	.46	10.40
Queen	62.40	6.80	.110	.22	.002	1.40	.86	.72	10.65
Regent	62.30	7.27	.048	.23	.008	1.95	.33	.09	10.50
Republic, Specular	67.62	2.28	.034	Trace	.018	.68	.30	.09	1.31
" Kingston	64.78	5.28	.036	.09	.030	1.30	.86	.16	1.31
Salisbury	62.10	4.91	.113	.30	.017	1.42	.36	.19	12.12
" Bessemer	63.04	5.80	.049	.21	.022	1.83	.21	.40	11.93
" Silica	49.31	24.10	.054	.36	.016	2.20	.35	.18	11.10
Savoy	62.05	7.75	.078	.34	.025	1.85	.27	.17	1.82
Section 21, Hematite	62.00	5.25	.130	.40	.021	1.10	.22	.21	9.48
Sheffield	62.08	.....	.018	.....	.....	.....	.....	.....	8.50
Swanzey	62.30	5.96	.077	.38	.045	1.89	.76	.42	13.00
Tilden Silica	45.40	33.10	.039	.....	.....	1.00	.....	.....	3.00
Volunteer	57.79	12.46	.069	.20	.019	2.23	.29	1.73	1.50
Winthrop	61.25	6.88	.135	.71	.027	1.34	.44	.88	11.37

Complete Analyses of Cargo Samples of Ore from the Menominee Range, Michigan and Wisconsin.

Name of Grade of Ore.	Iron.	Silica.	Phos.	Mang.	Sulp.	Alum.	Lime.	Magn.	Moist.
Appleton	63.30	4.61	.018	.27	.019	1.30	.52	.47	8.50
Aragon	63.60	3.25	.080	.10	.011	.98	.20	.11	6.75
Badger	59.20	4.83	.131	.42	.110	2.76	1.10	2.89	9.07
Claire	54.46	3.65	1.21	1.26	.015	1.13	8.20	1.22	6.60
Clifford	41.87	37.80	.011	.10	.012	1.00	.35	.34	2.14
Crystal Falls	58.55	4.25	.721	.20	.008	1.16	2.64	.77	7.20
Davidson	56.06	6.33	.170	.37	.101	3.41	1.54	2.97	8.40
Dunn	58.61	3.88	.573	.58	.033	1.88	1.80	.83	8.70
Elmwood	58.10	5.91	.172	.42	.096	3.06	1.03	2.38	9.05
Florence	57.60	4.67	.343	.....	.214	2.65	1.20	1.53	7.59
Grenada	63.78	4.30	.092	.23	.003	1.34	.35	1.05	6.70
Great Western	61.03	3.98	.249	.92	.052	2.31	.92	1.65	6.50
Hemlock	60.20	4.71	.309	.39	.010	2 81	1.87	1.32	6.62
Hiawatha	61.60	3.65	.130	.27	.082	1.35	.93	.27	6.00
Homestead	54.87	4.68	.086	.38	.003	2.05	3.48	4.87	.....
Iron River	59.80	5.29	.356	.37	.028	3.41	.69	.37	9.50
Keel Ridge	40.64	37.42	.046	.20	.006	.90	1.35	1.00	2.90
Lincoln	60.86	3.86	.24	.45	.010	1.67	2.14	1.73	6.50
Loretto	60.50	8.82	.017	.29	.077	1.92	.15	.47	8.69
Ludington	65.60	.....	.024	.....	.....	.....	.....	.....	7.06
Mansfield	63.36	4.14	.044	.38	.019	1.97	.46	.23	11.00
Mastodon	61.00	4.50	.350	.30	.075	2.75	.50	.30	9.00
Millie	62.10	3.20	.027	.20	.030	1.25	1.08	1.05	5.47
Paint River	56.40	3.48	.502	.378	.068	6.12	5.58	.718	8.00
Pewabic	64.36	3.85	.009	.19	.003	.48	1.10	1.35	6.46
Pewabic Genoa	42.79	35.28	.007	.10	.....	.96	.52	.95	4.32
Rex	57.68	5.65	.063	.81	.016	1.21	1.48	4.39	6.53
San Jose	65.41	3.23	.013	.24	.043	1.24	.16	.25	7.60
Sheridan	58.50	6.50	.130	.50	.250	4.00	.60	.70	9.12
Star Ludington	63.50	2.50	.086	.40	.010	1.00	.50	.40	7.00
Toledo	53.55	18.23	.010	.18	.003	.65	1.20	1.57	6.28
Tyrone	62.00	3.91	.106	.19	.008	.72	1.50	1.90	.....

Complete Analyses of Cargo Samples of Ore from the Gogebic Range, Michigan and Wisconsin.

Name of Grade of Ore.	Iron.	Silica.	Phos.	Mang.	Sulp.	Alum.	Lime.	Magn.	Moist.
Anvil	62.74	4.09	.055	.82	.018	.92	.47	.11	12.86
" East Vein	61.00	5.25	.058	2.18	.014	.86	.05	Trace	12.75
Atlantic	62.27	4.59	.040	1.11	.023	1.15	.18	.07	10.48
Ashland	62.91	4.32	.036	.21	.021	1.92	.12	.14	10.70
Aurora	62.94	4.02	.029	.48	.022	1.10	.37	.01	9.68
Brotherton	62.60	7.05	.037	.46	.004	1.10	.30	.10	11.11
Colby	56.80	8.20	.071	5.75	.010	.98	.24	.18	9.30
Cromwell	60.00	3.75	.032	3.10	.010	.75	.10	.13	9.58
Eureka	61.65	6.75	.066	1.40	.011	.95	.52	.22	9.80
Iron Belt	62.54	5.05	.042	.54	.015	1.65	.10	.02	10.67
Lawrence	60 40	8.90	.057	.45	.023	1.30	.09	.02	9.72
Melrose	63.07	2.96	.029	1.08	.016	.78	.10	.08	9.11
Montreal	64.88	4.10	.042	.33	.025	1.02	.10	.05	8.57
Newport	55.70	8.30	.040	7.40	.016	.98	.30	.07	9.44
Norrie	63.18	3.62	.041	.34	.017	1.74	.19	.13	9.86
Palms	63.50	3.62	.045	.88	.013	1.11	.16	.10	11.58
Pearce	58.60	11.44	.062	.64	.014	.81	.22	.13	7.60
Pence	65.20	5.50	.019	.24	.014	.28	.15	.05	.....
Rand	59.62	5.25	.033	3.21	.017	1.07	.66	.60	9.15
Shores	65.23	4.48	.029	.29	.....	.55	.....	.....	7.50
Tilden	63.56	3.97	.043	.78	.009	1.38	.22	.21	12.46
Windsor	64.00	4.00	.045	.63	.008	.80	.31	.10	.....

Record of Ore-Docks on the Great Lakes.

Railway.	Location.	Dock Number.	Length of Dock.	Width of Dock.	Height of Dock (Water to Deck).	Number of Pockets.	Storage Capacity. (Gross Tons.)	Height of Water to Center of Hinge-Hole.	Length of Spouts.	Angle of Bottom of Pockets.	Remarks.
			Feet.	Ft. In.	Ft. In.			Ft. In.	Ft. In.		
D. & I. R. R. R.	Two Harbors, Minn.	1	1056	41 0	45 6	141	18,000	26 10	23 0	{ 84 x 37°30′ 57 x 32°40′	
		2	1056	41 0	46 6	176	23,900	27 9	23 0	{ 98 x 37°30′ 78 x 32°40′	
		3	540	49 0	51 6	90	16,000	28 10	23 0	38°40′	
		4	1008	49 0	51 6	168	30,000	28 10	23 0	38°40′	
		5	1008	49 0	54 0	168	33,000	30 10	25 0	38°40′	
D., M. & N. R. Ry.	Duluth, Minn.	1	2304	52 0	53 8½	384	57,600	30 0	27 9	45°00′	
		2	1152	52 0	57 8½	192	34,560	32 0	27 9	45°00′	
D., S. & W. R. R.	Allouez Bay, Superior, Wis.	1	600	49 8	52 0	100	15,000	29 9	26 0	43°21′	Old part.
			900	49 8	57 0	150	25,500	33 0	27 8	45°00′	New extension.
C. & N. W. Ry.	Ashland, Wis.	1	1404	46 8	54 0	234	36,036	32 10	27 0	39°30′	
		2	1404	46 0	45 0	234	24,156	23 10	27 0	{ 210 x 42°00′ 24 x 45°00′	} 81 feet 9 inches height of hinge-hole.
	Escanaba, Mich.	1	1104	37 0	48 6	184	24,104	28 10	21 0	39°30′	
		2	1152	37 0	40 3	192	20,928	24 1	21 0	31°26′	
		3	1356	37 0	52 8	226	30,284	31 2	27 0	45°00′	
		4	1500	37 0	48 6	250	32,750	28 10	21 0	37°53′	
		5	1392	37 0	53 3	232	43,152	28 6	21 8	40°00′	
D., S. S. & A. Ry.	Marquette, Mich.	1	1700	40 0	45 0	270	27,000	25 0	.....	39°00′	
		3	1200	53 6	37 0	213	12,780	23 0	.....	.....	
		4	1200	36 8	47 3	200	28,000	27 9	21 1	39°45′	
L. S. & I. Ry.	Marquette, Mich.	1	1200	52 0	54 0	200	36,000	30 9	24 7	38°45′	
M., St. P. & S. Ste. M. Ry.	Gladstone, Mich.	1	768	37 0	47 0	120	15,000	26 8	21 8	40°00′	
Wis. Central Lines.	Ashland, Wis.	1	1908	36 0	54 6	314	33,500	30 4	27 0	{ 234 x 42°30′ 80 x 50°45′	

Members and Associates Elected.

The following persons were elected as members or associates by mail-ballot, November, 1897:

Members.

H. Foster Bain,	Des Moines, Iowa.
Thomas J. Barbour,	San Francisco, Cal.
Charles Lowthian Bell,	Middlesborough, England.
Arthur Hatfield Sumner Bird,	Salt Lake City, Utah.
Robert Macdonald Bird,	London, England.
John Blatchford,	Terry, So Dak.
William McC. Cameron,	Leadville, Colo.
Luis Campa,	Guanajuato, Mexico.
Hugo Carlsson,	Elyria, Ohio.
John S. Carnahan,	Sombrerete, Mexico.
Louis B. Carr,	Ouray, Colo.
Frederick H. Clymer,	Rockwood, Tenn.
Charles Frederick Courtney,	Broken Hill, New South Wales.
Sydney Cullingworth,	Cue, W. Australia.
William H. Davies,	Hazleton, Pa.
John Race Godfrey,	Sydney, New South Wales.
James Barrett Goodwillie,	Columbus, Ohio.
C. H. Hamilton,	Winnipeg, Manitoba.
Abbot A. Hanks,	San Francisco, Cal.
M. P. Gentry Hillman,	Birmingham, Ala.
James B. Laughlin,	Pittsburgh, Pa.
Frederick Ledoux,	Paris, France.
James I. Long,	Parral, Mexico.
Francis Arthur Malins,	Cripple Creek, Colo.
Arthur Leggett Neale,	Spitzkop P. O., Lydenburg District, So. African Rep.
Martin Nesbitt,	Chihuahua, Mexico.
Christopher Guy Orme,	Dublin, Ireland.
Harold Thomas Power,	Michigan Bluff, Cal.
Richard W. Rodda,	Terry, So. Dak.
J. S. Schultze,	Boonton, N. J.
Harry R. Skinner,	Roodepoort, So. African Rep.
Thomas Eddy Thomas,	Coolgardie, W. Australia.
Charles R. L. Tucker,	Chihuahua, Mexico.
Arthur W. Warwick,	Wickes, Mont.
Floyd Weed,	Cyanide, Colo.
Frank G. Willis,	Cripple Creek, Colo.
Robert A. Wood,	London, England.
Lewis Thompson Wright,	Keswick, Cal.

Associates.

Arthur Vincent Corry,	Golden, Colo.
E. McGowan,	Newcastle-on-Tyne, England.
F. D. Murray,	Newcastle-on-Tyne, England.
C. F. Rand,	New York City.

Change of Status.

John Gross,	Elyria, Colo.
John S. Pechin,	Cleveland, Ohio.

PAPERS.

The Handling of Material at the Blast-Furnace.

BY AXEL SAHLIN, SPARROWS POINT, MD.

(Chicago Meeting, February, 1897.)

THE old charcoal-furnace, with its square stone stack, perched on the hillside at the bank of the stream; the water-driven blowing-engine, with its wooden connecting-rod, and noisy leather flap-valves; the inclined bridge, up which the wheelbarrow wended its way to the open tunnel-head, where the artist top-filler, who alone possessed the secret of how to stock the furnace, presided, are pleasing memories from my childhood and my Swedish home. There was the frame-built coal-shed, where crude basket-work sleds gathered during the short winter days from the silent forest to discharge their loads of charcoal; there was the pile of dark magnetic ore, replenished from other sleds; there were the piles of pig-iron, with their cabalistic trade-marks, stacked on the shore of the lake, awaiting the coming of spring and open water. Everything was peaceful, calm and deliberate; 25 to 50 tons of iron per week was a fair average output.

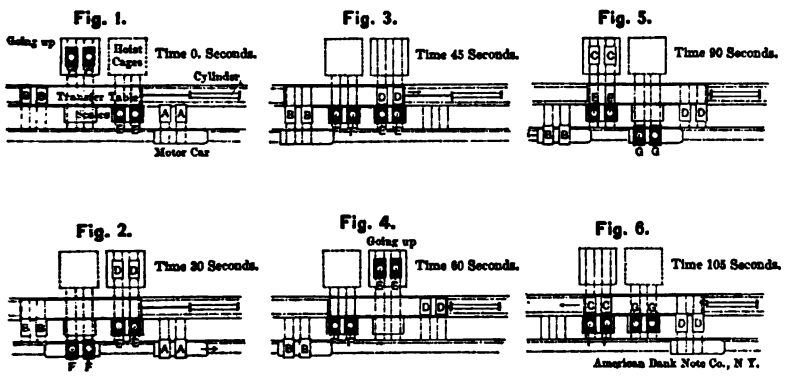
I cannot refrain from sketching this pen-picture of a generation ago, as a contrast to the furnace-plants and problems with which we of the present day have to deal. The evolution of the blast-furnace, especially the American blast-furnace, during the last third of a century has indeed been radical, making the question of getting the material to the furnace and the product away from it promptly, cheaply and regularly—the problem once satisfactorily solved by the cart or sled, the wheelbarrow and manual labor—one of great difficulty and grave importance.

It is my purpose in this paper briefly to record some of the most important steps which have been taken to solve this problem, and to offer a few suggestions as to what might next be done to keep up with, and anticipate, the increasing producing capacity of the blast-furnace.

The spread of the railroad net-work in all iron-producing districts early introduced the freight-car and the car-load as the natural unit in the transportation and handling of stocks; and, except where the furnaces are located directly on navigable water-fronts, the yards and stock-houses are arranged with a view of handling such cars to the best advantage. The elevated track or trestle is therefore a universal feature of the stock-yards. It was a natural step to build, between these trestles and under the tracks, bunkers or hoppers, with funnel-shaped bottoms, from which the stock could be drawn without shoveling. These hoppers are great labor-savers, if rightly proportioned and provided with efficient drawing-doors, but for years past, the success experienced with them has generally been indifferent; the reason being, that the shape of the hoppers often was governed, not by the laws of gravity and friction, as it should be, but rather by the proportions of the existing trestles, without much regard for the mechanical necessities of head-room, pitch, etc.

An interesting installation of hopper-storage is in use at the furnace-plant of the Dowlais Iron Company, at Cardiff, South Wales. The plant consists of four furnaces with a stated capacity in 1895 of 200 to 225 tons per day, each. Behind the furnaces, and on lines parallel with them, are located two rows of upright cylindrical sheet-metal tanks, about 30 feet in diameter and 60 feet high, resting on columns. The lower part of these hoppers is contracted and provided with chutes for drawing the stock, as required, direct into the charging-bogies. The flow of stock through each chute is regulated by a sliding door, entering the chute from the under side through a slot extending from side to side across the bottom of the chute. This device works perfectly, and is worthy of attention. Over the top of each row of tanks runs a standard-gauge track. At each end of these tracks is built a powerful hoist, operated by water-filled counterweights. At one end, the loaded wagons are raised to the level of the overhead-track, where a special engine hauls them over the proper tank. The wagon is discharged by dropping the bottom. It is then moved to the opposite end of the overhead track, whence it is lowered to the yard-level by a hoist similar to that at the raising-end. It must be borne in mind, that the four-wheeled English goods-wagon has a capacity

of not above 10 tons, and is therefore better adapted than the American 60,000-pound freight-car to this method of handling. The same height and construction of tank is used for handling coke, as well as flux and ore. It was reported, that, after the coke-tanks were once filled, the abrasion and crushing of the coke was very insignificant. By means of this apparatus, five men on the turn were able to stock each furnace. The men were paid by the ton of product, and were said to earn from eight to nine shillings per day. It is evident that the storage-capacity under this system is limited, and it would hardly be available where a winter supply of ore must be laid in, or a large reserve of stock for other reasons must be kept at the furnace.



System of Charging, Maryland Steel Company.

In plants requiring a large storage-capacity, the trestles are often extended to a considerable distance from the furnace. The expense of hauling an increasing number of charging-bogies grows in proportion to this distance, and would be prohibitory without improved transfer-arrangements.

The Maryland Steel Company, of Sparrows Point, Md., has overcome the difficulty of a long transfer by employing electric trolley-cars. Behind and parallel to the four furnace-stacks is located the stock-house, a magnificent steel structure 1180 feet long and 100 feet wide, containing four lines of tracks, supported on trestles, 20 feet high. The trestles are continued some 600 feet beyond the end of the stock-house; the extension being used exclusively for the storage of foreign ore. The hoists are of the ordinary upright reciprocating type, with plat-

forms 12 feet 6 inches by 12 feet 6 inches, served by Crane hoisting-engines with 14-inch by 16-inch cylinders. Along the front of the stock-house is located a narrow-gauge, depressed trolley-track, extending beyond the extreme end of the trestles. Over the track is suspended the trolley-wire, carrying a 250-volt current. On this track travel a number of motor-cars, each driven by a 6 H. P. motor and carrying two transverse tracks, level with the floor under the trestles, and sufficiently long to receive the wheels of the charging-cars, which, by a simple locking-arrangement handled by the operators, are scotched in proper position. The motion of the transfer-cars is perfectly controlled from a cab or stand at the rear end of the car. The operator becomes in a short time so expert, that he is able to stop his car within one-half inch of the point desired. Under each trestle, or bin, are two parallel tracks at right angles to the stock-tracks, corresponding with those on the transfer-cars. In front of the hoist are placed the scales, of the Fairbanks six-beam type, but with the levers and edges placed overhead, and the scale-platforms suspended underneath—an arrangement which facilitates inspection and repairs, and brings the delicate working-parts of the scales away from the dust. Each scale- and hoist-platform carries two tracks corresponding to those on the transfer-cars. Between the scales and the hoist-cages is placed a depressed transfer-table operated by blast-pressure, supporting six transverse tracks, so spaced that, at either end of the stroke, four of them correspond to those on the scales and hoist-platforms, while the remaining two match return-tracks for the empty cars. The system of working the charging-cars is explained by Figs. 1 to 6, which can be understood by following the changed position of the cars marked with the same letter.

The loaded cars are marked black; the empty cars are outlined; the arrows indicate commencing motion. One weigher with two assistants and two motor boys on each (twelve hours') turn have, with this apparatus, without difficulty, handled stock for a furnace producing nearly 10,000 tons of iron in one month, and with a record of 392 tons in twenty-four hours. At times the ore is brought in to the scales from a distance of 900 feet. Two transfer-cars are used for each furnace. The construction of charging-cars is shown by Figs. 7, 8 and 9. They are pro-

vided with drop-bottoms, held by a simple and very effective toggle-lock. The coke-car has a capacity of 55 cubic feet or

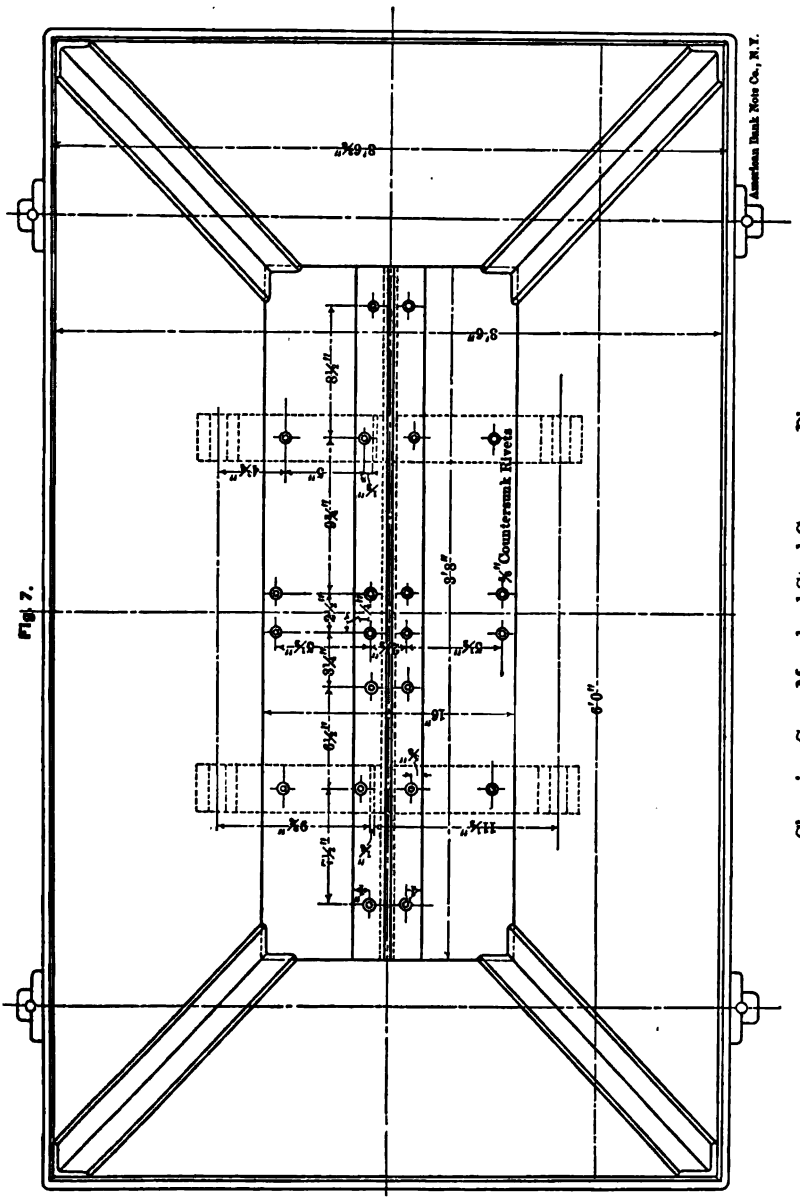


Fig. 7. Charging-Cars, Maryland Steel Company, Plan.

1680 pounds. The ore- and limestone-car has a capacity of 30 cubic feet, or about 2240 pounds. The cycle of operation, including the hoisting, discharge and return to motor-cars of four

charging-cars, requires about two to two and one-half minutes. For handling the heavy charging-cars, the top platform is fitted with tracks running across the hopper on either side of the bell-rod. The two cars raised on the same hoist are run, one behind the other, upon one of these tracks; the bottoms are dropped by springing the toggle-locks; a special air-cylinder lifts the far end of the track, and the cars are sent by gravity back to the hoist-cage. Such was the arrangement used on the first furnace equipped for this charging-method. When the furnace was blown out it was found, however, that the continuous dropping of the stock at four points only had caused the furnace-walls to cut so, that the cross-section of the upper part of the shaft was nearer square than circular. The system had, however, given such good satisfaction and economical results, that it was decided to overcome this difficulty by placing the tracks spanning the hopper on a turn-table, revolving through an angle of 45° and worked by air-pressure. The present arrangement of the top of "A" furnace is shown by Figs. 10 and 11. The cars constituting every alternate charge are revolved by means of the table, so that stock is dropped in eight places instead of four, insuring better distribution and decided economy in fuel. To make sure that the table is regularly revolved, a recording instrument, inaccessible to the top-fillers, is attached to the operating-cylinder. Three men on each turn do the work on the top of the furnace.

Since the days of the inclined foot-bridge and the wheelbarrow, the improvements in the stock-hoist have kept pace with the efficiency of the furnace proper. As a first step, rails and a platform-car, carrying the wheelbarrows, or, later, the charging-bogies, were placed on the inclined bridge, water, run into a counterweight-tank, furnishing power. One such bridge often served two or three furnaces. Later, hydraulic or pneumatic pressure took the place of the water acting by weight. The inclined bridge gave way to the vertical hoist-tower of the present day.

In 1860 Mr. John Fritz designed and built for the Bethlehem Iron Co. a number of pneumatic hoists, which are yet in operation, and which, so far as I know, have not been described in the *Transactions* of the Institute. These hoists were a striking improvement on anything then in use, and the principle

employed in them may yet be utilized, in connection with an air-compressor and higher pressure, in designing a modern

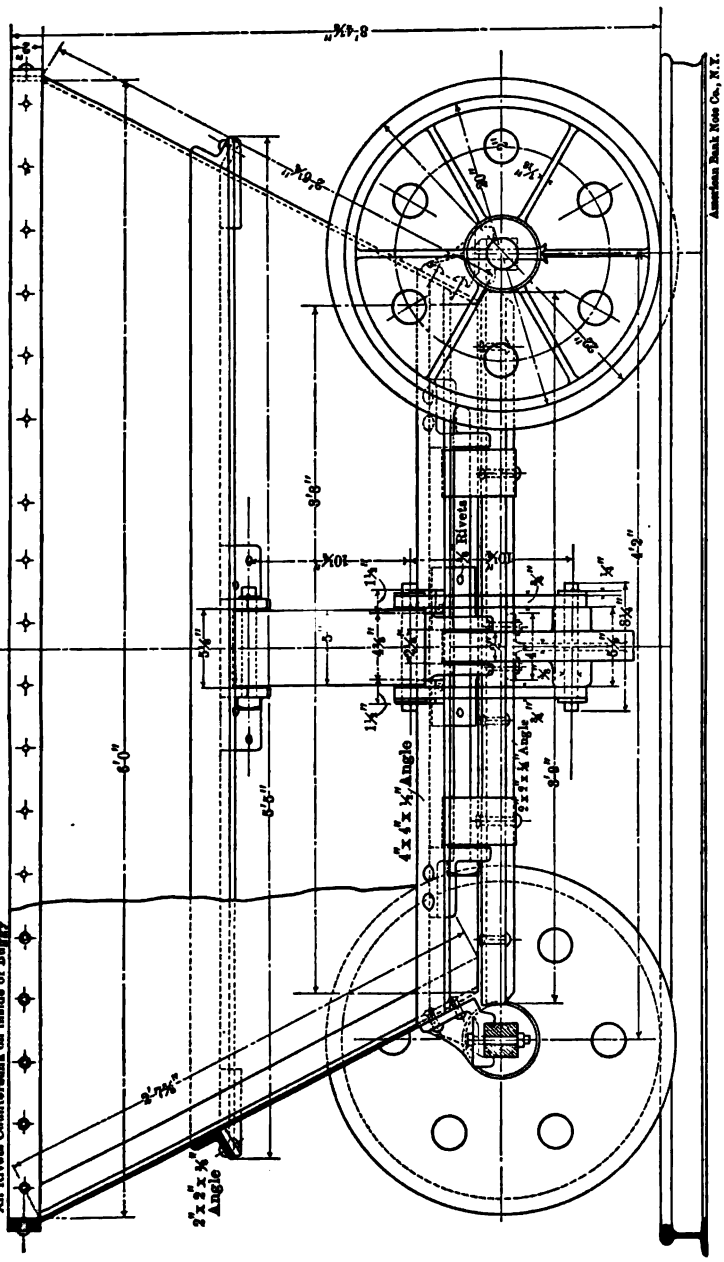


Fig. 8. Charging-Cars, Maryland Steel Company, Longitudinal Section.

rapid-moving hoist of large capacity. The arrangement of the hoist is shown by Fig. 12.

The single cage is placed between two cast-iron air-cylinders, extending the entire height of the hoist-tower. It is guided by rollers traveling on V-shaped rails, cast in one with the cylinders. In the bore of these work leather-packed pistons, to each of which are attached two hoist-ropes. These ropes run through the upper head of the cylinders over sheaves, supported by the top-frame of the tower, the opposite ends being secured to the hoist-cage. When cold-blast pressure is admitted to the cylinders above the pistons, the load is raised; when the air is exhausted the platform is lowered. A hoist, built on the same principle, but with one central air-cylinder, was installed at the Freemansburg furnace of the Bethlehem Iron Co. It is shown in Fig. 13.

Such a hoist, slightly modified, was also built in 1871 by the late Mr. George Fritz for Furnace No. 5 of the Cambria Iron Co., Johnstown, Pa. As late as in 1885 this hoist was working side by side with a double-cage steam-hoist at Furnace No. 6, the latter being equipped with an engine of good make (as hoisting-engines were built in 1878). The air-hoist gave far less trouble and caused fewer delays than the steam-hoist. It had, however, no spare capacity, and both hoists were soon afterwards altered and equipped with modern engines.

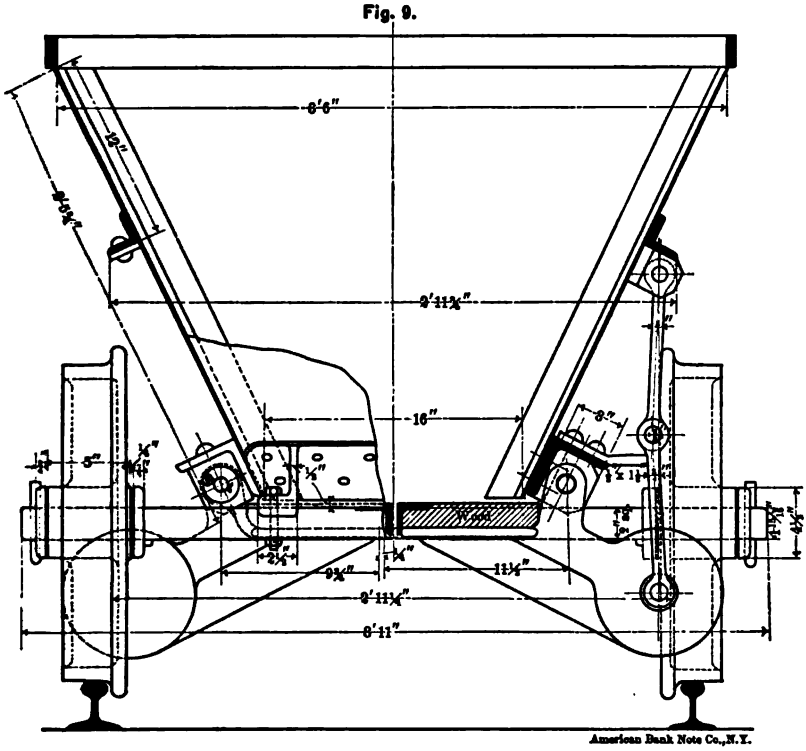
The successful blast-furnace hoisting-engine is of comparatively recent date. In 1872 the Crane Elevator Company, or rather their predecessors, put their first engines in operation at the Joliet Steel Works, the Vulcan Steel Works of East St. Louis, the Spearman Iron Company's furnace and some other plants. These engines were belted. Since that day the hoisting-engine has been strengthened throughout, perfected and enlarged to meet the growing requirements. Steam-engines had, it is true, been used for blast-furnace hoists earlier, but they were crude, deficient in reliable stop- and safety-appliances and did not gain or deserve general confidence.

Great credit is due to the engineers who have evolved the present compact, faithful, ever-ready hoisting-engine, without which furnace-managers would hardly succeed in making the wonderful records, with which they are now astonishing the technical world.

But we have gradually entered on a new departure. The vertical hoist, if not the hoisting-engine, has reached its limit.

It is difficult to see how, without extravagant use of labor, more than from 1500 to 1600 tons of material per twenty-four hours can be brought in hand-bogies to the hoist-platforms, lifted 100 feet or more, removed from the platforms, dumped into an open hopper and the empty bogies returned to the stock-house. These operations, incessantly repeated with the utmost promptness, become a task almost beyond the endurance of man, and so we are returning again, in blast-furnace construction, to the

Fig. 9.



Charging-Cars, Maryland Steel Company, Cross Section.

inclined plane—now, however, made to carry larger cars or skips, and designed to raise a considerable portion of a charge on each trip. The real or assumed difficulty of obtaining proper distribution in the hopper has for years been made an objection to this method of elevating the stock. It was, however, in use at the Lucy furnaces in Pittsburgh as early as 1881, and has been in use at the Lebanon furnaces, Lebanon, Pa., since 1885. In most of the recently built furnace-plants

the skip-hoist is made a feature, and the labor of the top-fillers has been dispensed with. By means of a double hopper, into the upper part of which the skips are emptied, the stock is undoubtedly distributed with sufficient regularity.

The skip-hoist at the Lebanon furnaces was designed by the Weimer Machine Company. It consists of a single inclined track, 7 feet 7 inches gauge, 301 feet long, with a rise from the stock-house floor to the tunnel-head of 95 feet. The track crosses the hopper above the depressed bell-beam, from which the conical bell is suspended by a short link only. On the track travels a drop-bottom tank 7 feet 2 inches in diameter, 7 feet 1 inch deep, with a capacity of 8000 pounds of stock. Recently the original drop-bottom doors have been replaced with a bell, whereby the capacity of the tank is somewhat increased and a better distribution secured. The 6-foot diameter hoisting-drum is carried on cast-iron brackets, bolted to the top of the furnace-shell. It is geared 6 to 1 to a duplex, vertical hoisting-engine (16-inch diameter of cylinders, 20-inch stroke) which is bolted to the furnace-shell below the hoisting-drum. The skip makes a trip in two minutes. The hoist is satisfactory, and has been duplicated at the furnace of the Duluth Iron and Steel Company, Duluth, Minn.

Through the courtesy of Mr. Edwin Thomas I am enabled to give some data regarding the Thomas patent furnace-filling apparatus, which is in successful operation at the Pioneer furnaces, at Thomas, Ala. This contrivance permits the hoisting of narrow-gauge drop-bottom cars to the top of the furnace and discharging them automatically. Each filling-car has a capacity of about 2000 pounds of coke or 4500 pounds of ore. The stock-house, 70 feet wide and 650 feet long, is located 150 feet from the furnace. The stock is unloaded from trestles in the usual manner. The filling-cars travel on tracks of 30-inch gauge, extending the entire length of the stock-house and having turn-outs and side-tracks running into the different stalls. At the center of the stock-house is a turn-table for swinging the cars on to the scale-track, whence they continue to the foot of the inclined hoist-plane. This is 190 feet long and has a pitch of 25°. It is composed of two spans of 7-foot deep lattice-girders, spaced 7 feet, center to center, and designed to carry a live load of 20,000 pounds. The girders are supported at the foot by a

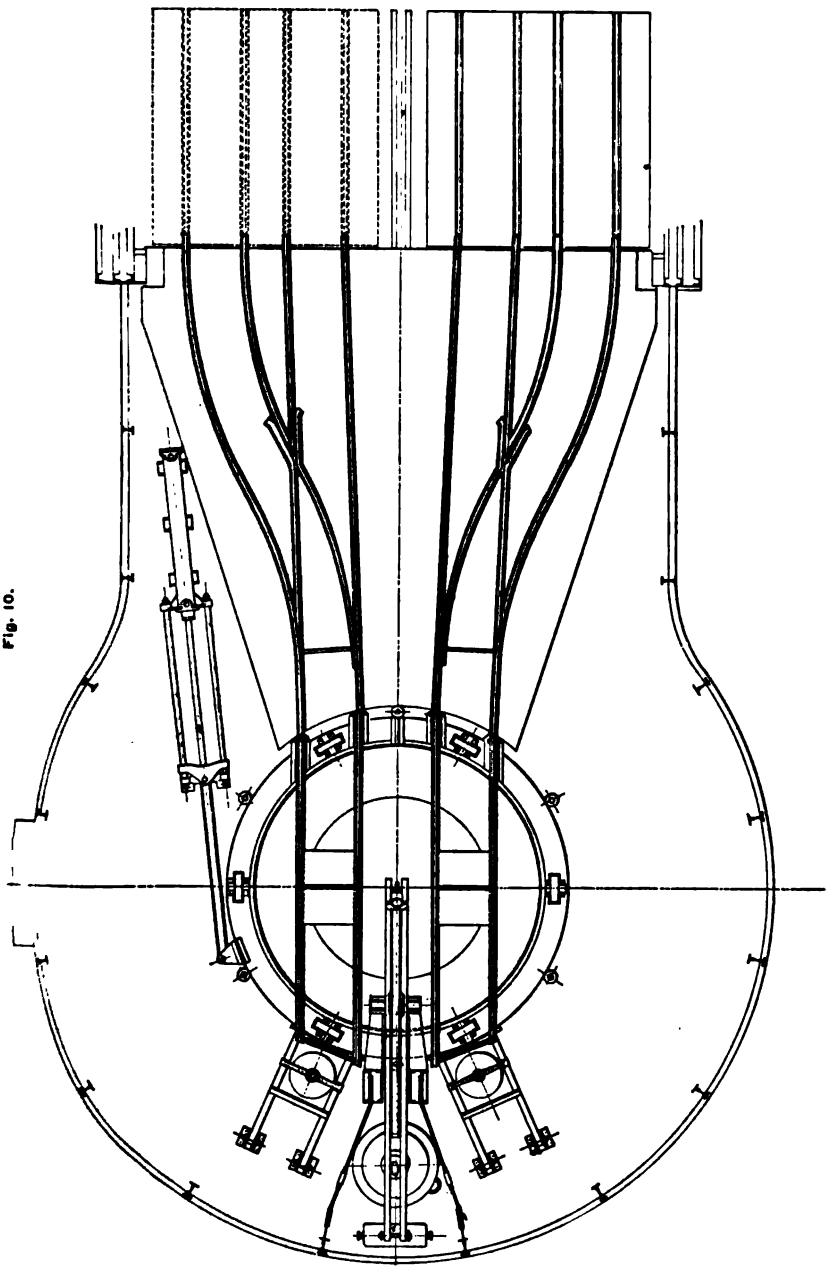


Fig. 10.

Top of "A" Furnace, Maryland Steel Company, Plan.

masonry pier, in the center by a steel structure and at the top by heavy brackets riveted to the furnace-shell. To the top-flange of the hoist-girders are bolted cross-ties, which support

a single four-rail track and a stairway. The gauge of the two outside rails is 30 inches, and on these run the filling-cars. The inside rails are gauged 22 inches. They support a special truck, permanently attached to the two ¾-inch hoisting-cables, and designed to push the loaded filling-cars ahead of it to the top of the incline. The outside rails connect with the tracks to the stock-house and scales, while the inside rails continue to descend below the general stock-house level, forming between them a pit or well, into which the pushing-car sinks, permitting the empty filling-car to continue over its top on the track, back to the stock-house. The loaded filling-car is, in its order, brought from the scales across the pit containing the pushing-car, which is made to re-ascend behind it, carrying it to the tunnel-head.

Fig. 14 shows the two cars traveling up the incline. The ingenious automatic dumping arrangement at the tunnel-head is shown in Fig. 15. The 30-inch track extends across the hopper. From the girders supporting this track is suspended an upper cast-iron hopper or funnel, with a 24-inch circular aperture concentric with the apex of the bell, which is 8 feet 6 inches in diameter and supported by four rods. The filling-cars are fitted with hinged drop-bottom doors, controlled by chains, levers and counter-weights in the shape of revolving rollers attached to the levers. As the car mounts, the counter-weights are raised by being brought to bear against curved tracks or matrices shown in the figure. The bottom-doors drop when the weights are lifted, and the stock is discharged into the upper cone of the hopper. When the car returns towards the incline the weights are released and the bottom-doors close automatically. The pushing-truck is not counter-balanced. To insure against any delay from break-downs the hoist is fitted with two engines (15- by 18-inch cylinders). The furnaces are of moderate dimensions, being 75 feet high, with a bosh-diameter of 16 feet 6 inches. The largest amount of stock required to be raised in one week has been 5500 tons. This has been handled by the hoist without crowding, and with considerable capacity to spare. This apparatus is undoubtedly economical in first outlay, clever in design and efficient in performance.

Recently double-track, alternating inclines with self-dumping skips and intermediate hoppers have been designed and

built which will have a large capacity, but which I am not at liberty to discuss at the present time. It is to be hoped that a description of the filling-arrangement at the new Duquesne furnaces will soon be placed before the Institute, as the recent

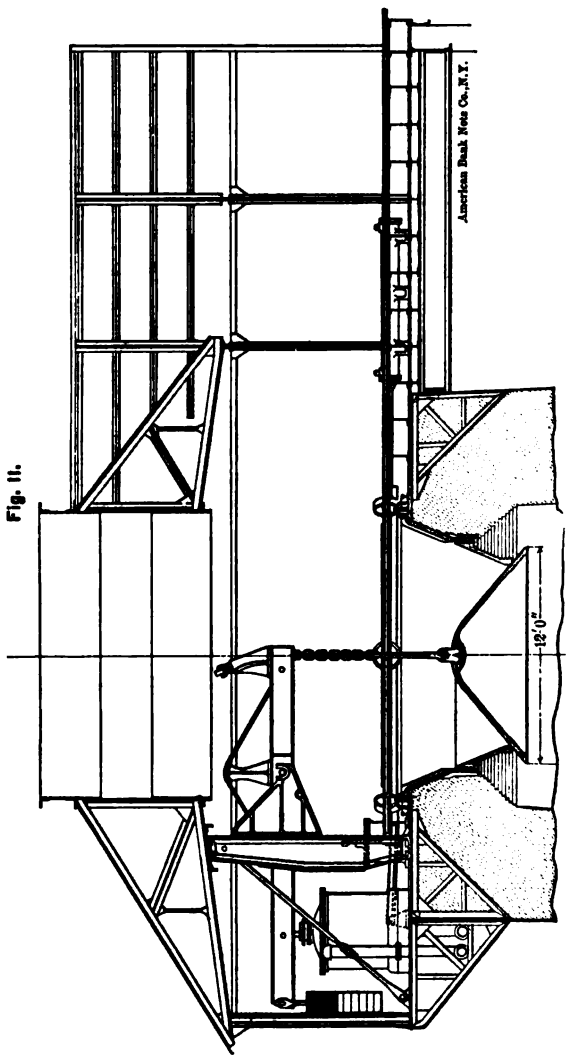


Fig. 11.

Top of "A" Furnace, Maryland Steel Company, Vertical Section.

performance of this arrangement widely surpasses anything attempted at any other blast-furnace.

At furnace-plants located on navigable water-fronts we find the Brown hoisting- and conveying-apparatus universally adopted

for unloading vessels and conveying the ore to the place of storage. The construction of one of the most recent and effective of these machines is shown in Fig. 16, showing side-elevation and plan of one revolving machine, and Figs. 17 and 18, showing a plant of sixteen machines, as built at the works of the Illinois Steel Co. at South Chicago. Its mode of operation is too well known and understood to require special description. The great advantage of this device, as compared with ordinary derricks, cars and trestles, in the rapidity of transfer, capacity of storage on a given area, low first cost of plant per ton of material stored, and in avoiding all rehandling, is obvious.

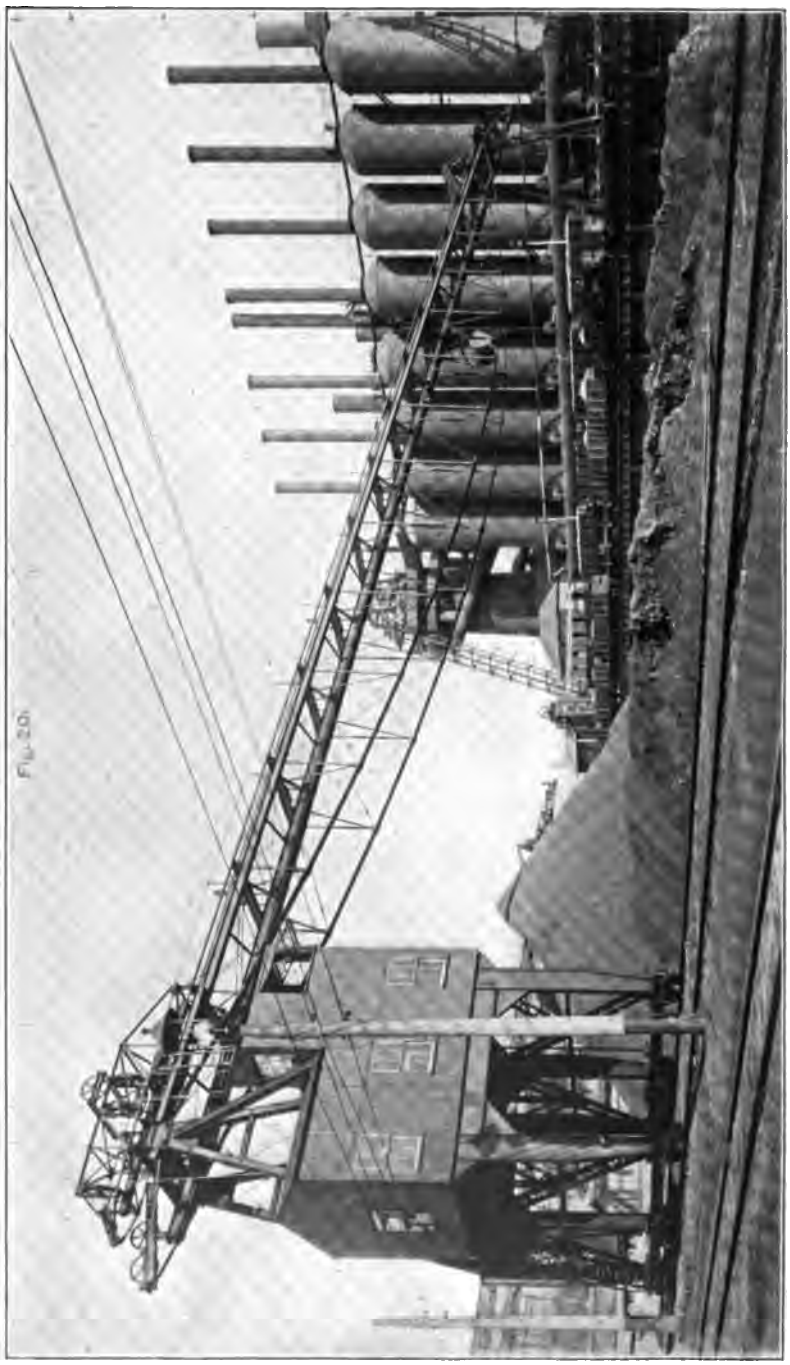
With one of these machines at the works of the Illinois Steel Co., South Chicago, between 500 and 600 tons of hematite-ore have been unloaded from steamer and stored at a point 280 feet from the ship in ten hours and by eight men. It is on record that on September 7th, 1894, the large steamer S. S. Curry, with 4569 gross tons of ore was unloaded at the above works in eleven hours, working nine hatches, that is with nine machines, with sixty-three men in vessel and twelve men operating the machinery. The Brown machine may be built as a gantry crane, spanning the stock-yard as well as in the ordinary cantilever form traveling or revolving. It is driven either by electricity or by steam.

The rope-tramway is frequently used around blast-furnace plants, especially on the continent of Europe, for long transfer of stock and for removing cinder. As it may be built in spans of 300 to 400 feet or more, it is especially valuable in crossing inaccessible areas. As instances, I refer to the installations long in use at Gutehoffnungshütte, Oberhausen, Germany, for transporting the cinder from the blast-furnaces to the dumps across the main line of the Berlin-Cologne Railroad, and to the cable carrying coal from the River Meuse to the extensive works of the Société John Cockerill, Seraing, Belgium.

One of the best applications of this system to blast-furnace plants in the United States may be seen at the works of the Buffalo Furnace Co., Buffalo, N. Y., where vessels bringing Lake Superior ores are unloaded at the rate of 2500 to 3000 tons a day, and the ore is distributed by wire tramways to different points in the stock-house, some 2000 feet away. In connection with quarries and mines, the system is suc-

Fig 19.

Brown Conveying-Apparatus, Duquesne Furnaces.



Brown Conveying-Apparatus, Duquesne Furnaces.

Brown Conveying-Apparatus, Duquesne Furnaces.

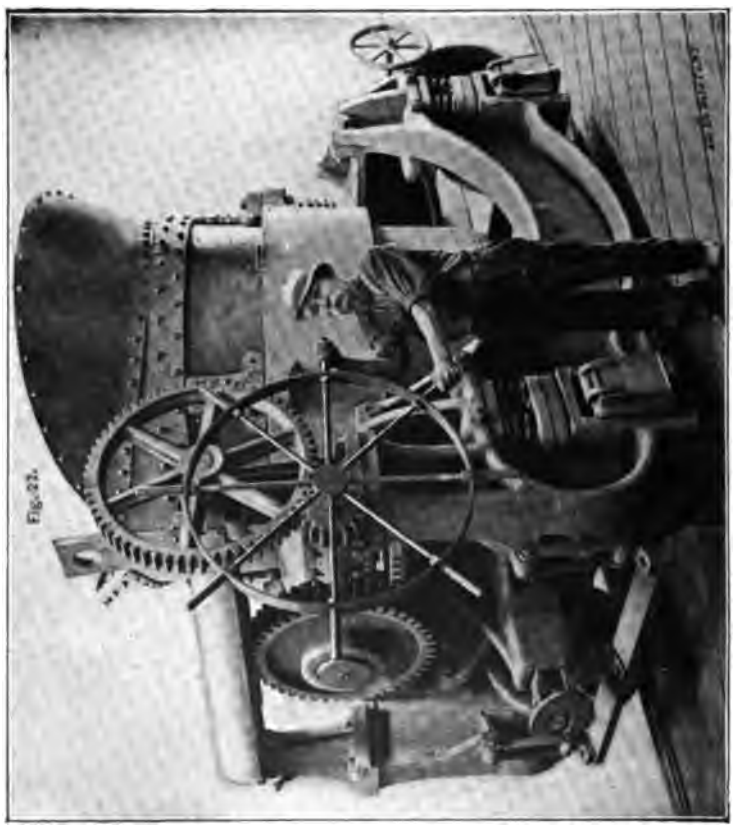
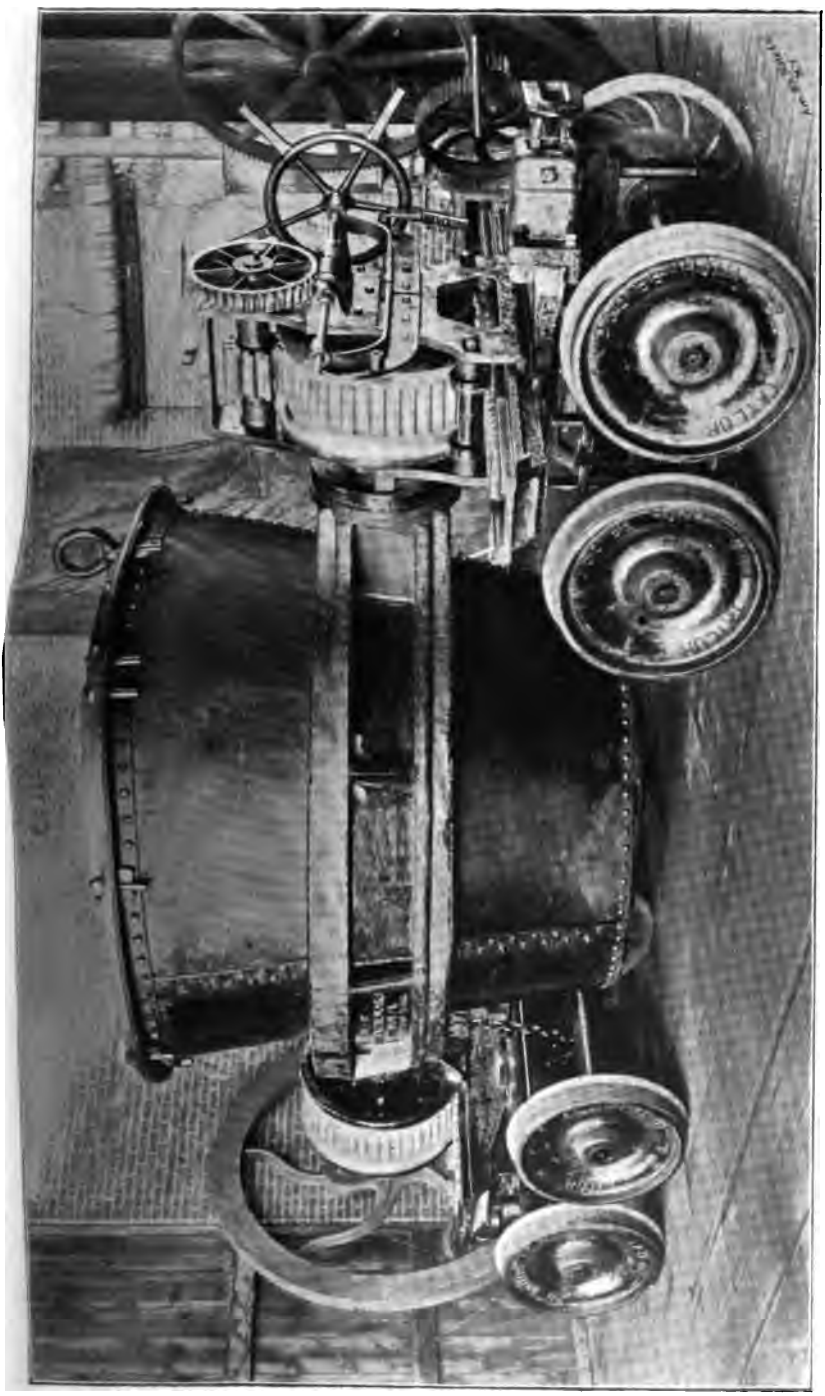


Fig. 22.

Transfer-Car and Ladle.



Weimer Cinder Car.

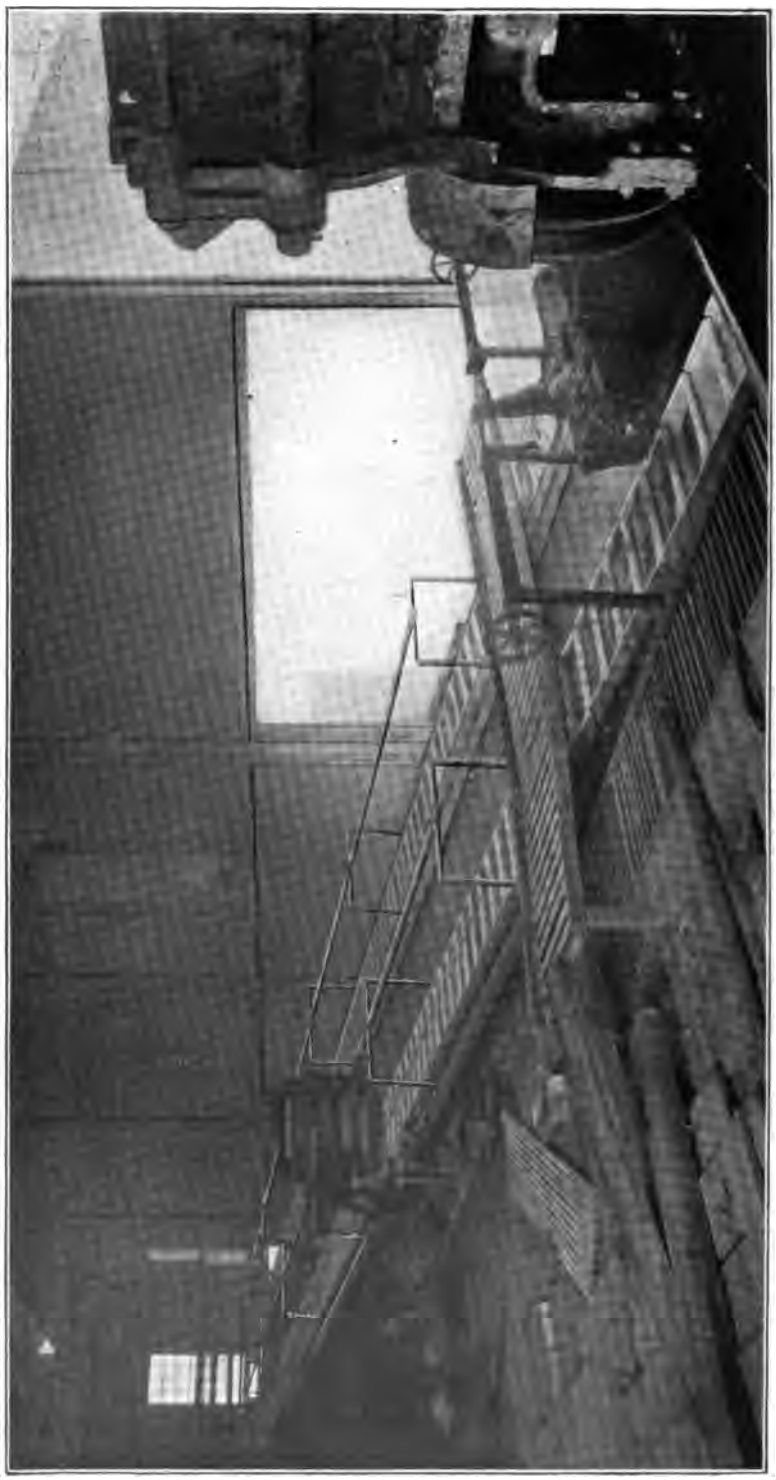


FIG. 29.

Uehling Casting-Machine.

FIG. 30.

Howden Slag-Conveyer; End Nearest Furnace.

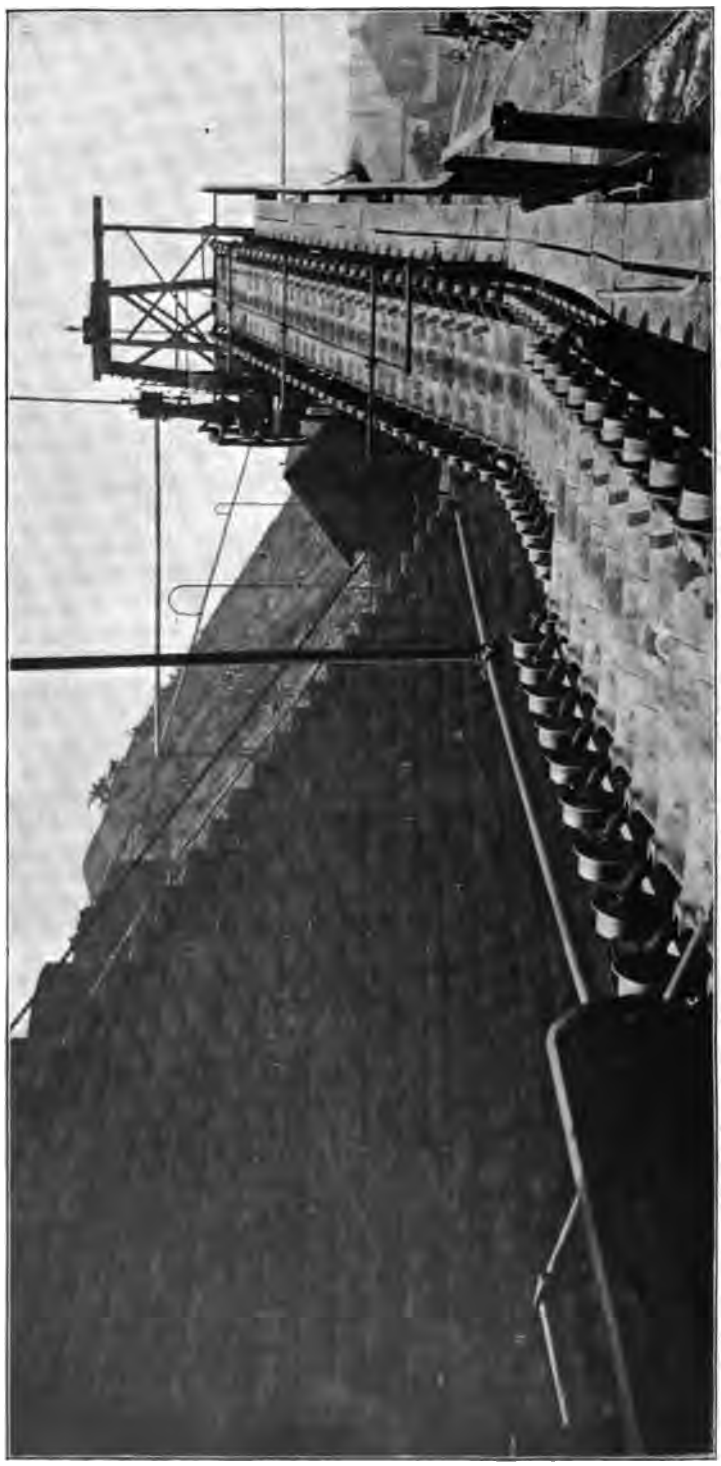


FIG. 31.

Howden Slag-Conveyer, as seen from Cast-House

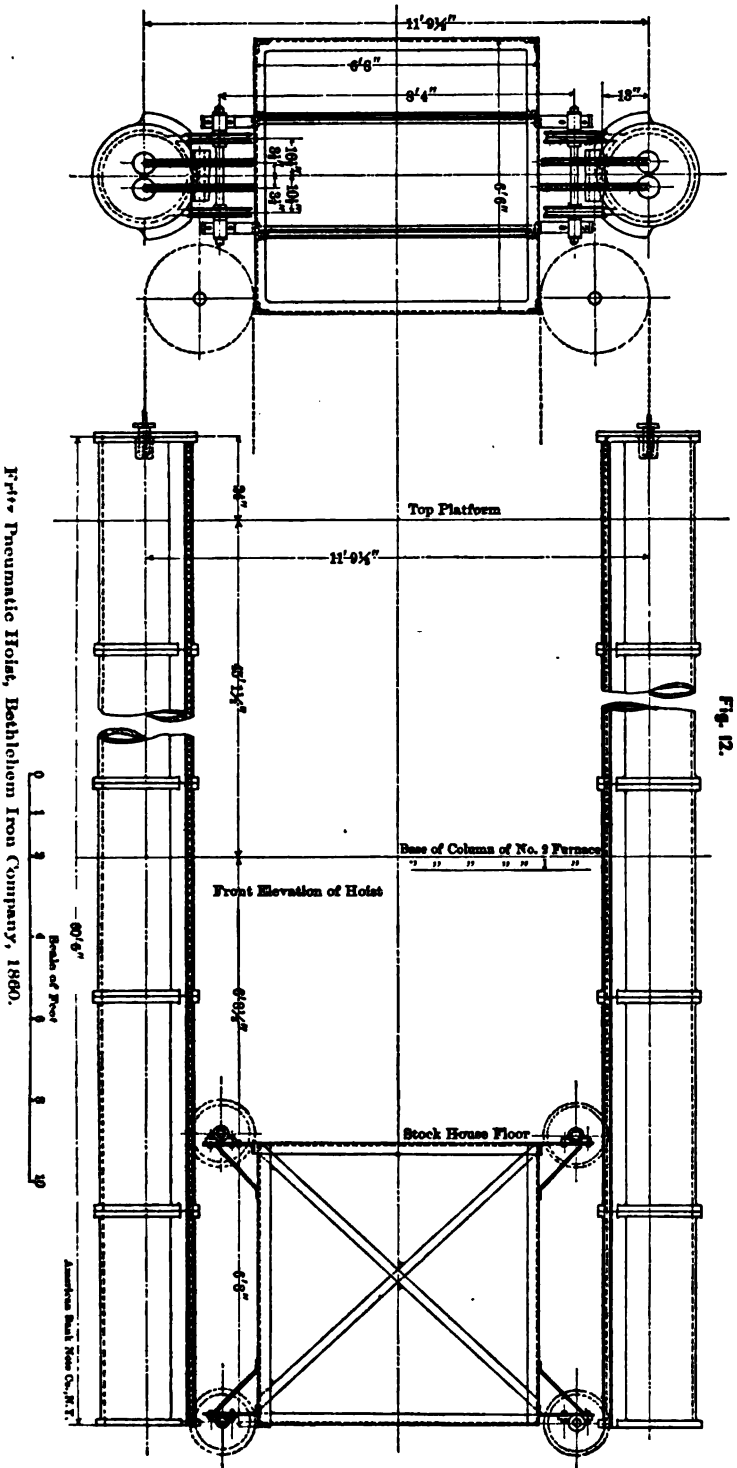
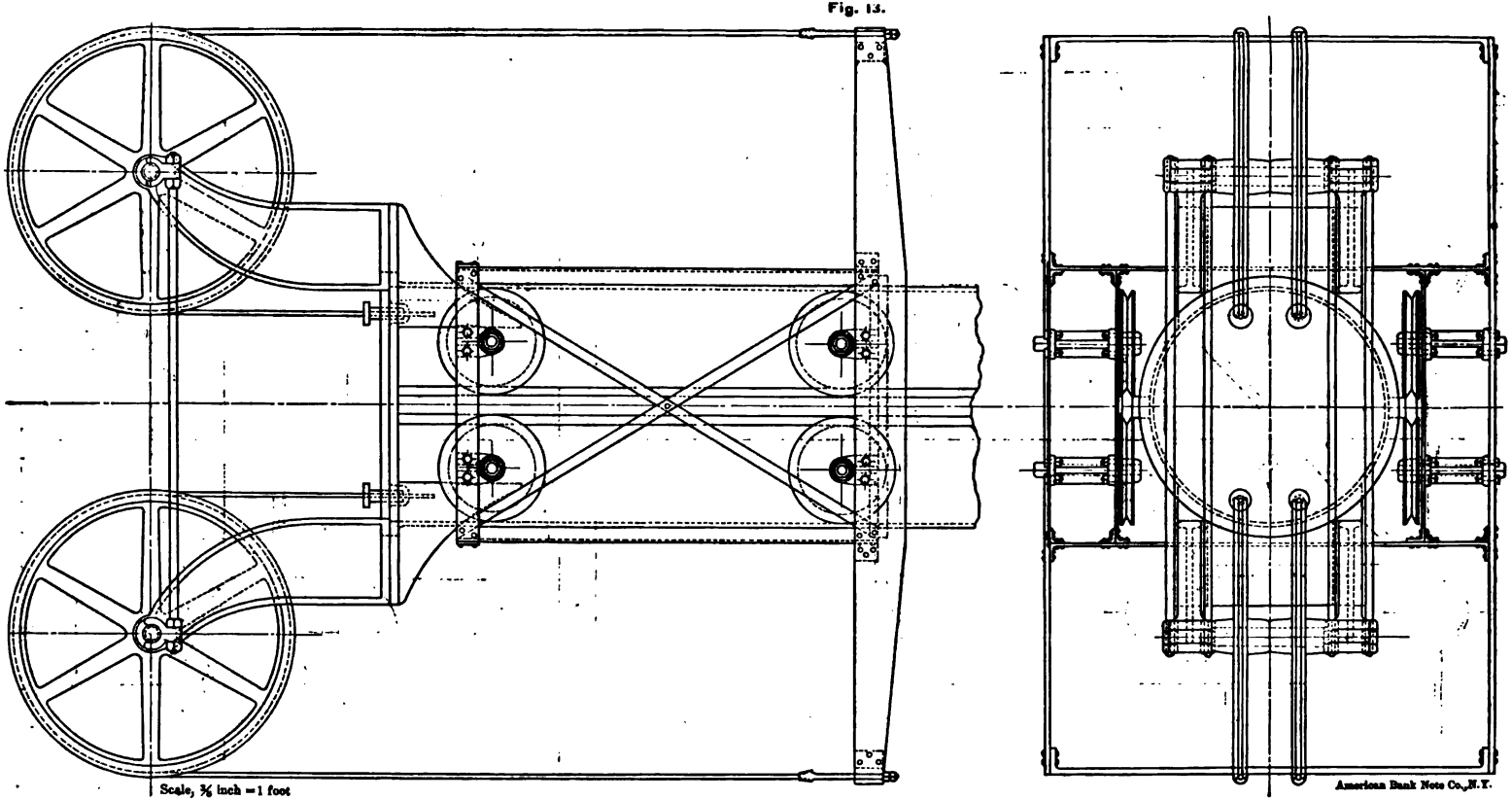


Fig. 12.

Fritz Pneumatic Hoist, Bethlehem Iron Company, 1860.

Fig. 13.



Pneumatic Hoist, Freemansburg.

cessfully operated at many places in this country, as well as in Europe.*

The manner of handling ore at the Duquesne furnaces of the Carnegie Steel Co., Ltd., is clearly shown by illustrations, Figs. 19, 20, and 21, for the use of which I am indebted to Mr. Alexander E. Brown, of the Brown Hoisting and Conveying Machine Co. The gantry cranes, or bridge-tramway machines, are operated by electricity. They have a clear span of 233 feet, covering the stock-yard, and a cantilever extension of 33 feet additional, covering the trestle-tracks. The bridges will clear a stock-pile 58 feet high. These cranes remove the ore from the cars, depositing it in the stock-yard, using for this purpose self-tipping ore-tubs; or, they carry the ore from the stock-pile to the charging-hoppers. For this work automatic shovels, designed and patented by the Brown Hoisting and Conveying Machine Co., take the place of the self-dumping ore-tubs shown in these illustrations. The working-capacity of each crane is from 1500 to 2000 tons of lake-ore per day of ten hours.

The facility with which power may be distributed to any part of the stock-yard or plant by means of electric motors, opens a new field for ingenuity in devising labor-saving appliances. All breaking and screening of material by manual labor should be a thing of the past. Electrically-driven conveying-belts, a good form of which is described by Mr. Thomas Robins, Jr., in the *Transactions* for 1896, vol. xxvi., p. 78, should take the place of the wheeling of materials from inaccessible portions of the yard. Electrically-driven slow-moving bucket-elevators may be placed anywhere around the plant without the old-time consideration for distance from steam-supply, and will handle and dispose of coal, coke-dust, ashes, flue-dust, sand or clay economically and promptly. The electric traveling-crane, which should cover every modern cast-house, will handle, at a nominal cost, heavy castings, such as bells, hopper-rings, hearth- and bosh-plates, iron- and cinder-troughs, chills, and other spare parts requiring periodical renewals. It will raise the worn cinder-cars from their trucks and replace them in a small fraction of the time generally consumed

* See *Trans.*, xix., 760, and xx., 766, for description of its construction and use.

for this operation. It will, if desired, remove the entire product in the form of pig-beds from the casting-floor, and may give invaluable assistance in cleaning up a wreck, break-out or boil of iron, and thus save the exasperating delay and disorganization of labor, with which all who have been practically engaged around a furnace-plant are so unpleasantly familiar.

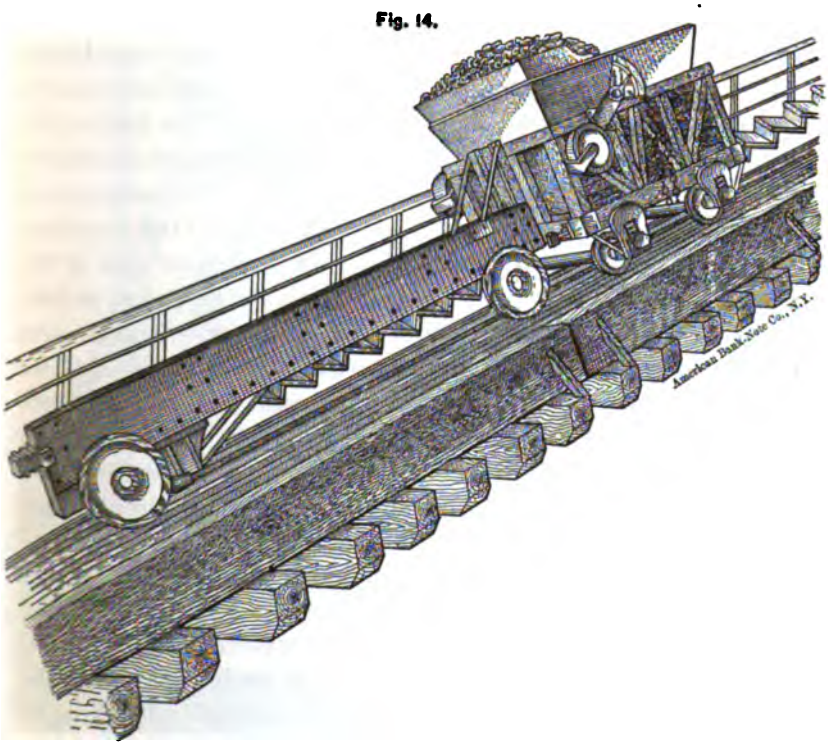
Comparing the methods for bringing the stock to the furnace tried at different plants, and the results obtained by them, it seems that a combination of some suitable form of the Brown hoisting- and conveying-machine, with traveling steam-shovels for reloading the ore, hopper-storage for coke and flux, electric transfer and a good form of the inclined skip-hoist, raising a large portion of the charge on each trip, would represent the most effective arrangement which the present state of the art will permit.

It is worthy of note, that while the stock-yard, storage, transfer arrangements and hoist, have been improved step by step, the cast-house, generally speaking, remains to-day in exactly the same condition as that of the old-time charcoal-furnace. As we have forced the furnace to an increased output, our only resource seems to have been to build additions to the cast-house, and make more frequent casts. But we still continue to break up the pig-beds with the crow-bar and hand-sledge; we carry the iron, piece by piece, from the casting-floor to the car by manual labor, we form the mould for receiving the iron anew for every cast, with a large expenditure of labor and loss of time, which we can hardly afford—all in the same manner as a generation ago. But it does not follow that this part of the furnace-plant has been neglected, but rather, that the difficulties encountered in devising practical improvements have been, at least until very recently, too great to be overcome.

Among the few improvements in the method of casting, the use of iron chills is the oldest. The chills save labor, especially while new, and facilitate the breaking of the pig-beds; but they are high in first cost, reduce the capacity of the casting-floor and close the grain of the iron. They will, therefore, never be used for foundry-iron, until our foundrymen consent to buy their metal on analysis instead of on fracture, as at present. The life of the chill is only from one to three years, depending partly upon the quality of the metal, partly on the

amount of patching and building up, that the furnace-manager is willing to have done before condemning a defective chill. It is, therefore, an open question, whether money is lost or made by casting in chills. In most cases a slight saving can probably be shown. For basic iron, chills are valuable, as they leave the pigs free from a siliceous coating of sand. When Sunday-iron only is handled in the cast-house, the saving by the use of chills is also more marked.

Fig. 14.



Thomas Furnace-Charging Apparatus.

The direct-metal process, or the running of the liquid iron into ladles to be carried to a receiver, or direct to the Bessemer converter, is a radical improvement in the method of disposing of the metal, but its scope is limited, as it can be used only in conjunction with a steel-plant; and even then the metal produced while the steel-works are idle, briefly named the Sunday-iron, must be taken care of at the furnace, for which purpose labor and casting-floor must be kept in readiness. It is often

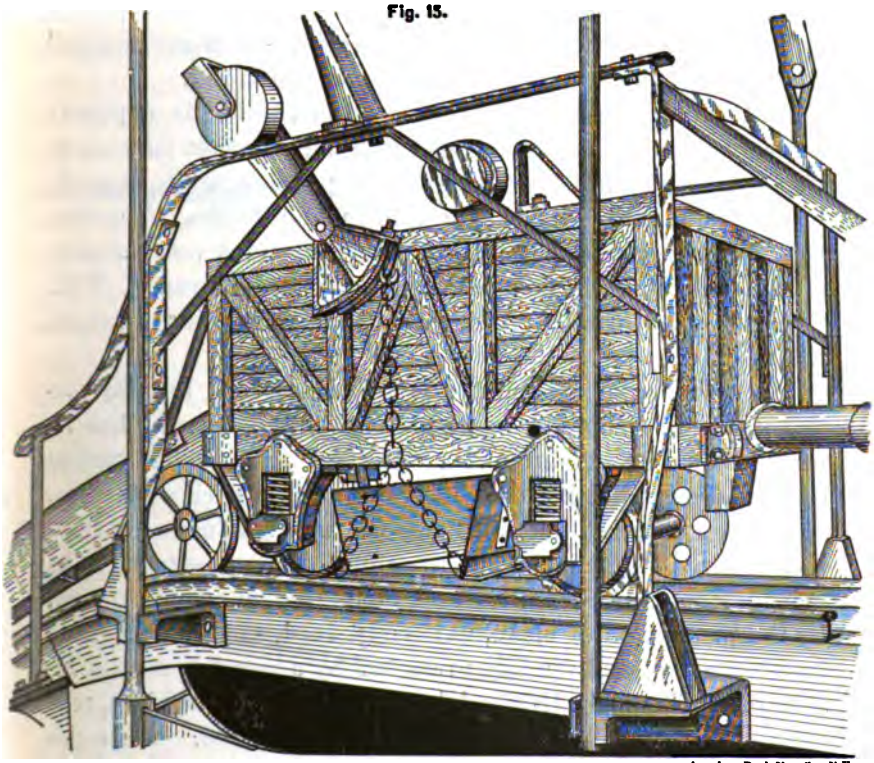
an awkward problem for the manager profitably to employ this reserve-force, which may be called on at any moment.

Fig. 22 shows an approved form of an 18-ton transfer-car and ladle, as used by the Maryland Steel Company and recently adopted by the Cambria Iron Company, the Cleveland Rolling Mill Company, and the Pennsylvania Steel Company.

The Dowlais Iron Company has introduced a device, invented by Messrs. Edward Evans and Enoch James, by means of which the handling of the metal is greatly simplified. The well-built furnace-plant at Cardiff consists, as above stated, of four furnace-stacks making Bessemer iron for the extensive acid open-hearth plant connected with the works. It may be considered as the exponent of the modern English furnace-plant. In front of the furnaces is located the uncovered casting-floor, intersected by curving, depressed cinder-tracks, which leave the casting-beds for each furnace in an irregular shape, unusual to the American eye. An elevated traveling-crane runs from one end of the plant to the other, on tracks parallel with the furnaces, and entirely covering the casting-floors. The pig-moulds and runners are formed in sand to wooden patterns. When the metal, cast into pig-beds, has cooled sufficiently to stand handling, the beds are successively lifted out of the sand by means of the traveling-crane and carried to the end of the track, where they are stacked or delivered directly to the pig-breaker, the principle of which is shown by the figure. It consists (see Figs. 23, 24, 25 and 26) of a heavy cast-iron frame *a*, the upper part of which carries four hydraulic cylinders: (1) The horizontal cylinder *l*, which works two pawls *k*, used for feeding the pig-beds to the breaker, and moving it the width of three pigs at each stroke; (2) the horizontal cylinder *o'*, which operates a clamping-block *m*, by means of wedges *n* (shown separately in detail), intended to hold the sow firmly against the lower part of the frame; (3) the vertical cylinder *b*, with the inverted plunger *c*, which carries three breaking-heads *e*, at different levels (this plunger successively detaches the three pigs from the sow and breaks them in the center); and (4) the vertical inverted cylinder *r*, the plunger of which breaks the overhanging portion of the sow. The vertical cylinders are counter-balanced, so that their plungers rise to their highest position, when pressure is cut off. The four cylinders are operated by one man. The bed is placed

by the overhead traveling-crane, or by an auxiliary jib-crane, with the front end resting on frame *a*, while the rear end is supported by truck *h*. The pawls feed it to the breaker until the entire bed is broken. The pieces of iron slide down an inclined apron *u*, to the wagon placed below the breaker. The machine does very good work. At the time of my visit only two furnaces were in operation, producing together about 400 to 450 tons of iron per day. It was reported that four men on

Fig. 15.



American Bank Note Co., N.Y.

Thomas Charging-Apparatus; Arrangements for Dumping at Tunnel Head.

each turn did all the work on the two casting-floors. Two men and two boys, on day-turn only, handled, broke and loaded the entire product. The handling of the iron in units of beds, weighing from 2½ to 3 tons, instead of pigs weighing from 70 to 120 pounds, is undoubtedly a step in the right direction.

As improvements in the method of dealing with the iron, two auxiliary machines for opening and closing the tap-hole deserve special mention.

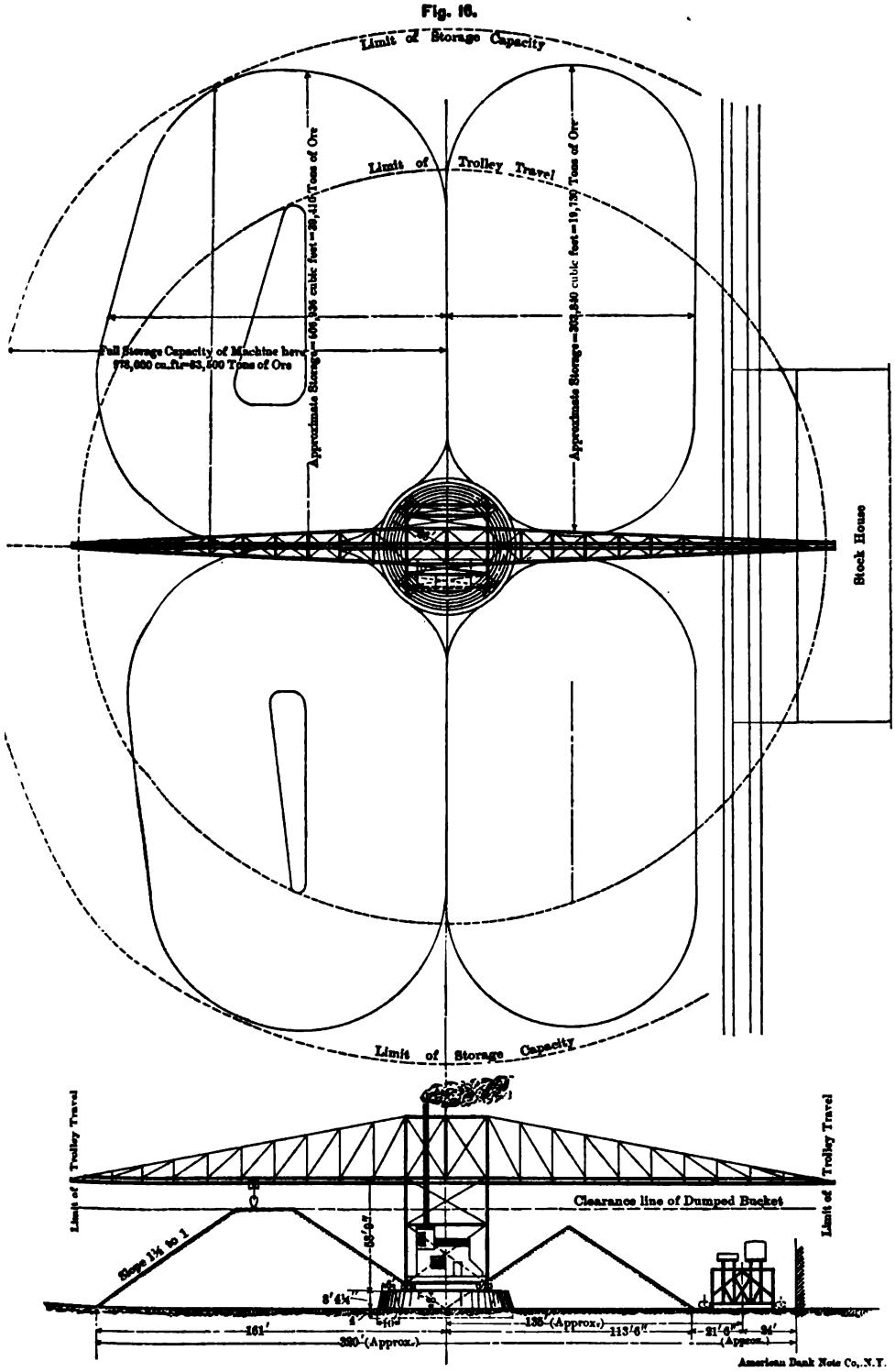
The tap-hole drill, a pneumatic rock-drill, guided by a strong frame and fed by a pneumatic cylinder, has been described by the inventor, Mr. David Baker, in the *Transactions* for 1892, vol. xxi., page 588. The tap-hole closing-machine, invented by Mr. Samuel Vaughen, is shown by Fig. 27, in the form in which it is used by the Maryland Steel Company. It consists of a pneumatic cylinder controlled by a plain side-valve. In front of the air-cylinder is bolted a second cylinder, which is filled with balled stuff or clay, such as is used for closing the tap-hole. The clay is forced out of the nozzle of the forward-cylinder and driven into the tap-hole by advancing a plunger fastened to the protruding end of the air-cylinder piston-rod.

Both the drill and the closing-machine, which is popularly known as the "gun," are swung from light cranes fastened to the furnace-columns, and are held in position, while at work, by suitable, easily detachable clamps and hooks. Together they have proved themselves time-savers, and a very valuable aid in controlling the flow of the metal from the furnace. With their aid, a cast can be made, without the blast being taken off the furnace.

The problem of designing a practical machine for the continuous or interrupted casting of blast-furnace metal has received considerable attention. Several patents for various contrivances, indicating the widely-felt need of such a machine, have been granted, but until a short time ago none of them had received a practical demonstration. The advantages to be gained, from an economical and metallurgical point of view, by the production of clean, uniformly-sized, sand-free pig-iron have been ably and forcibly presented by a member of the Institute, Mr. Henry D. Hibbard, in his paper on this subject, read before the British Iron and Steel Institute in 1896. Since this paper was published experiments have been in progress, resulting in the evolution of the Uehling machine for casting, conveying and automatically loading blast-furnace metal, which is now in successful operation at the Lucy furnaces, Pittsburgh, and, on the strength of its performance there, has been adopted to replace the system of sand-casting and pig-breaking hitherto used at the Duquesne furnaces.

The Uehling machine, Fig. 29, consists of two endless chains about 90 feet long from center to center of the end sprocket-

Fig. 16.

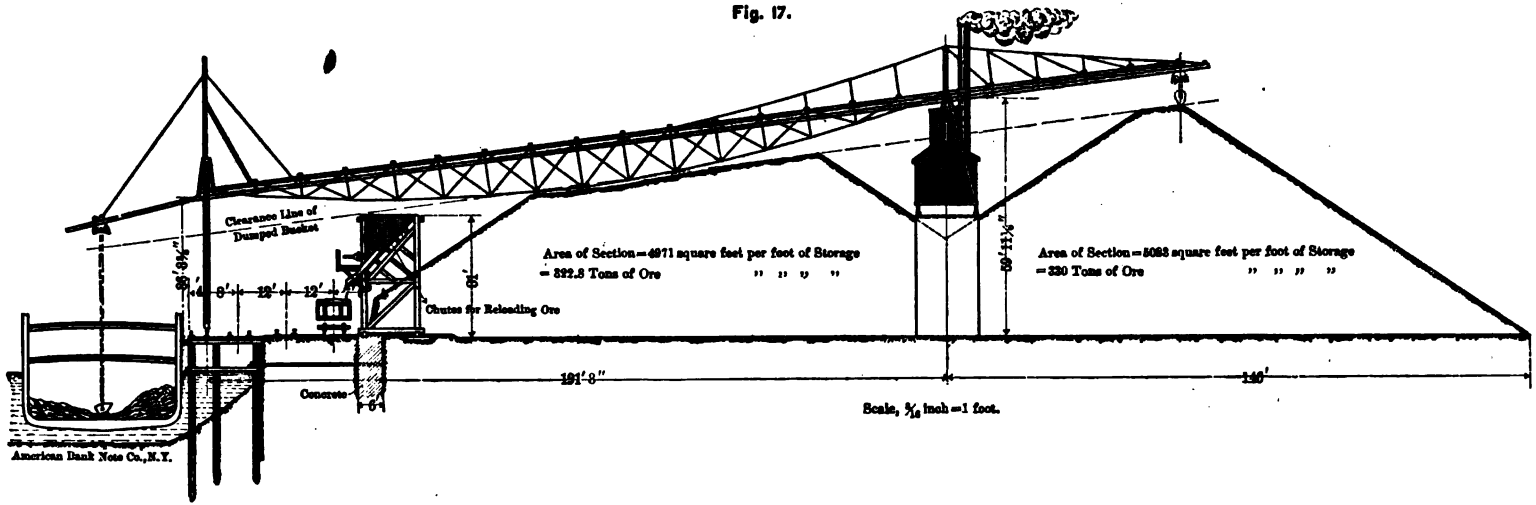


Brown Conveying-Apparatus, Plan and Elevation.

wheels. To the links of these chains are bolted a series of iron moulds or buckets, with edges overlapping one another, so as to form a continuous casting-platform, moving at a rate of about 15 feet per minute. The machines are worked in pairs, two lines of moulds traveling side by side. The iron is run from a ladle-car through a T-shaped trough into the slowly-advancing moulds. By tilting the trough one way or the other the flow into the two lines of moulds is regulated to a nicety. At the end of six minutes, when the filled moulds have reached the end of the conveyer, the pigs are sufficiently cooled to permit their being dropped from the casting-machine upon a conveyer, which carries them through a cooling-tank filled with water, and delivers them directly on the railroad-car ready for shipment. To promote the cooling of the pigs while in the moulds they are sprinkled with water, which is so regulated by a revolving distributer that the force of the water is directed at all times towards the center of the advancing pigs, while the moulds themselves are, in a measure, protected from the cooling action of the spray. The empty moulds, returning to the casting-trough, are coated with lime-wash, thrown against them in the form of an atomized spray, which insures an even and strongly adhesive coating of the inner surfaces of the moulds. The moulds are made of soft gray iron. They are 22 inches long, 12 inches wide, and 7 inches deep, and have a thickness of metal of $\frac{3}{4}$ inch. The life of a mould, so far as past experience indicates, is about 45 days of continuous work. One double casting-machine, as described above, and shown in Fig. 29, readily handles one ton of metal per minute. The largest quantity of iron cast in one single day has, at this writing, been 729 tons, which represents the daily output of the two Lucy furnaces.

The cooling-conveyer is built of $\frac{1}{2}$-inch steel plates riveted to two endless link-chains, forming a smooth platform, on which the yet red-hot, though solidified, pigs drop from the moulds. The carrier is depressed, passing through a cooling-tank filled with circulating water, during the passage through which the pigs are thoroughly cooled. The further end of the conveyer takes an upward course, and delivers the pigs directly upon the shipping-car, without any manual labor whatever. For the data given above I am indebted partly to Mr.

Fig. 17.



Brown Conveying-Apparatus, South Chicago, Elevation.

James Scott, superintendent of the Lucy furnaces, and partly to Mr. James W. Miller, special agent for Uehling's method for casting and conveying metal.

The importance of this new development and its effect on the arrangement of the future blast-furnace should not be overlooked or underrated. It would seem as if the metal should be run from the blast-furnace into a mixer or receiver heated with furnace-gas and of sufficient capacity to hold at least two casts. From this mixer it should be cast directly into the moulds of the casting-machine. With a sufficient receiver-capacity, we may safely dispense with the expensive cast-house and with the host of men employed therein, and at the same time we should obtain a more uniform metal, and may tap the furnace at any time we choose. From a metallurgical point of view the advantages thus secured are equally worthy of consideration.

Sand-free pig-iron is imperatively demanded by the basic open-hearth process. The sand adhering to the iron in the form of free silica is far more destructive to the basic lining than is the silicon chemically combined in the metal. For sand-free metal the limit specified for the amount of Si in the pig may therefore be raised, admitting for use many grades and makes of iron, which, if cast in sand, would be unfit for the basic open-hearth process. The same observation applies to iron used in the puddling process. In the Bessemer works and foundry, in short, wherever the iron is melted in a cupola, absence of sand, while not affecting the composition or metallurgical value of the metal, effects a marked saving in flux, in fuel, in iron-loss in the slag, and in the labor of bringing the considerable quantity of sand to the cupola and the slag away from it. One ton of pig-iron cast in sand weighs, commercially speaking, 2268 pounds, while 2240 pounds is the accepted weight of one ton of sand-free iron. It is now customary to pay a premium of about 25 cents per ton for iron cast in chills, over and above the price for the same iron cast in sand. Yet, the iron cast in chills is far from sand-free, as the chills are seldom perfect, being more or less patched with sand; and, besides, some sand is carried from the sand-runners and adheres to the iron. If metal cast in chills is at a premium, metal cast in the casting-machine, and therefore absolutely sand-free, should command a yet higher premium.

Many of our foundrymen, who are compelled to work without the aid of a laboratory, and who have been trained to judge

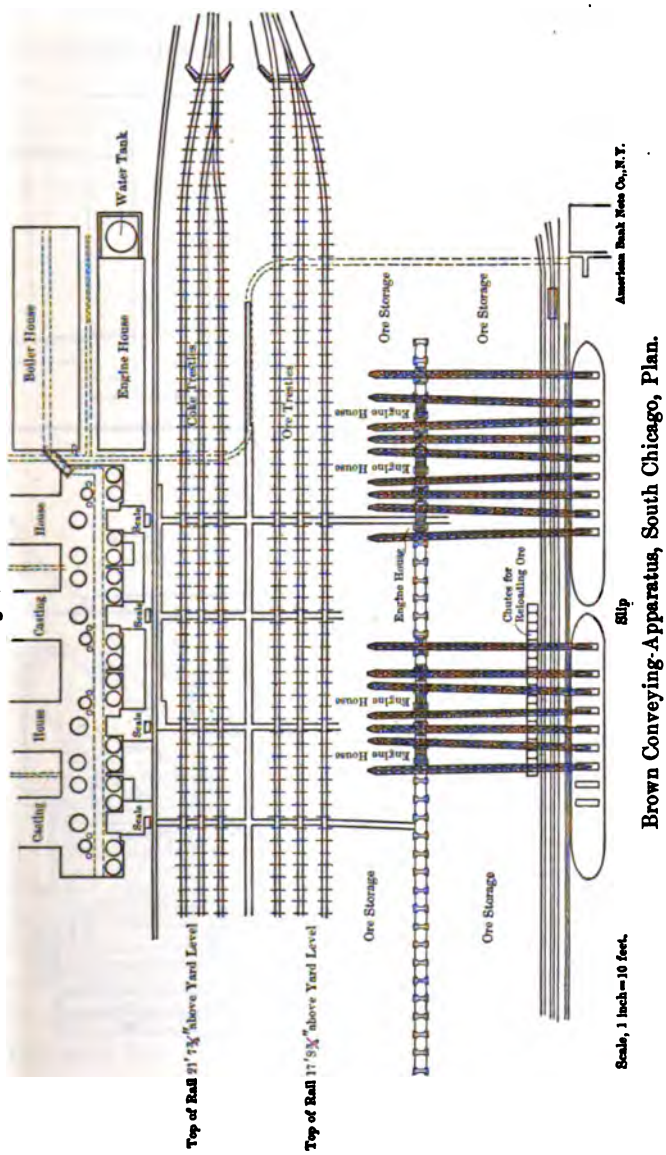


Fig. 18

Brown Conveying-Apparatus, South Chicago, Plan.

their iron and to make up their mixtures from the appearance of the open fracture, will undoubtedly object to the casting in chills, which deprives them of their accustomed guide for con-

ducting their operations. But if the foundryman would follow the example of the blast-furnace manager, who invariably judges his metal from the appearance of the fracture of a sample cast in an iron chill, he would find such a fracture more reliable and less variable than that of the slowly-cooled sand-pig.

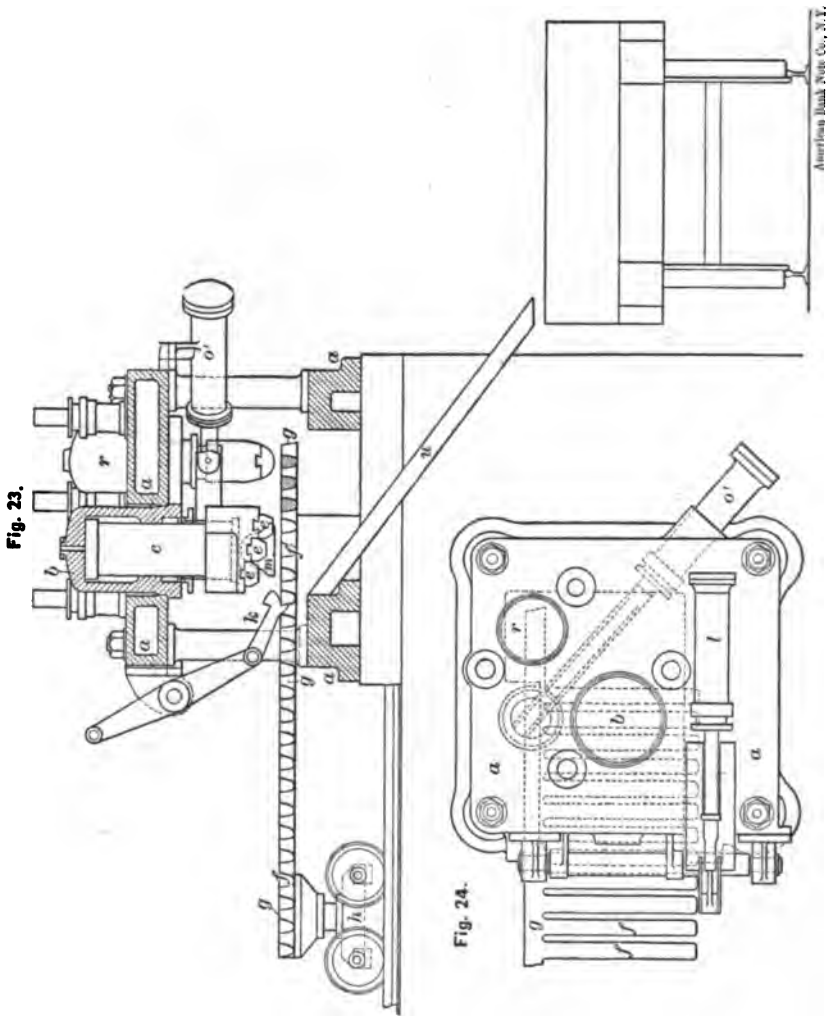


Fig. 23. Dowlais Pig-Breaker.

Fig. 24.

Besides, it is to be hoped that the progress of the iron-casting industry will be such, that before long the assistance of the chemist and the laboratory will be as helpful and indispensable in the foundry as it is to-day in the steel-plant or at the blast-furnace.

In dealing with the blast-furnace cinder, our methods to-day are nearly perfect. As late as in 1883 it was still customary at most blast-furnaces to form rough moulds in slag, on what used to be called the cinder-wharf. Into these the hot cinder was run. It was broken before cooling and loaded on dump-cars by manual labor, or it was formed into round cakes, often with an iron staple cast into the center of each cake, for convenience in handling. The cakes were lifted by a swinging crane and loaded on cars. Men were required to prepare the moulds and

Fig. 25.

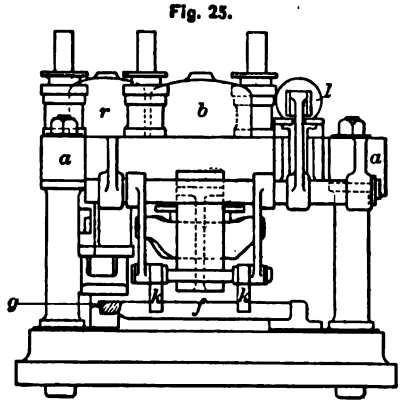
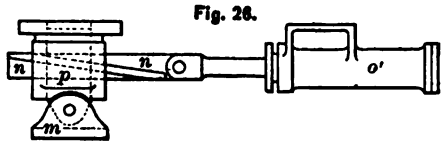


Fig. 26.



Dowlais Pig-Breaker.

runners, to load the cinder, and afterwards on the dump to unload the unwieldy masses. Cinder-cars with removable tops next came into use. The repairs to these cars were heavy, and considerable labor was required to remove and replace the tops, and to bar the cakes of cinder from the flat bottoms of the cars over the dump. Of late years the Weimer gondola cinder-car has been adopted by most progressive furnace-men.

The latest form of a 200-cubic-foot Weimer car is shown in Fig. 28. It is lined either with fire-brick or with cast-iron

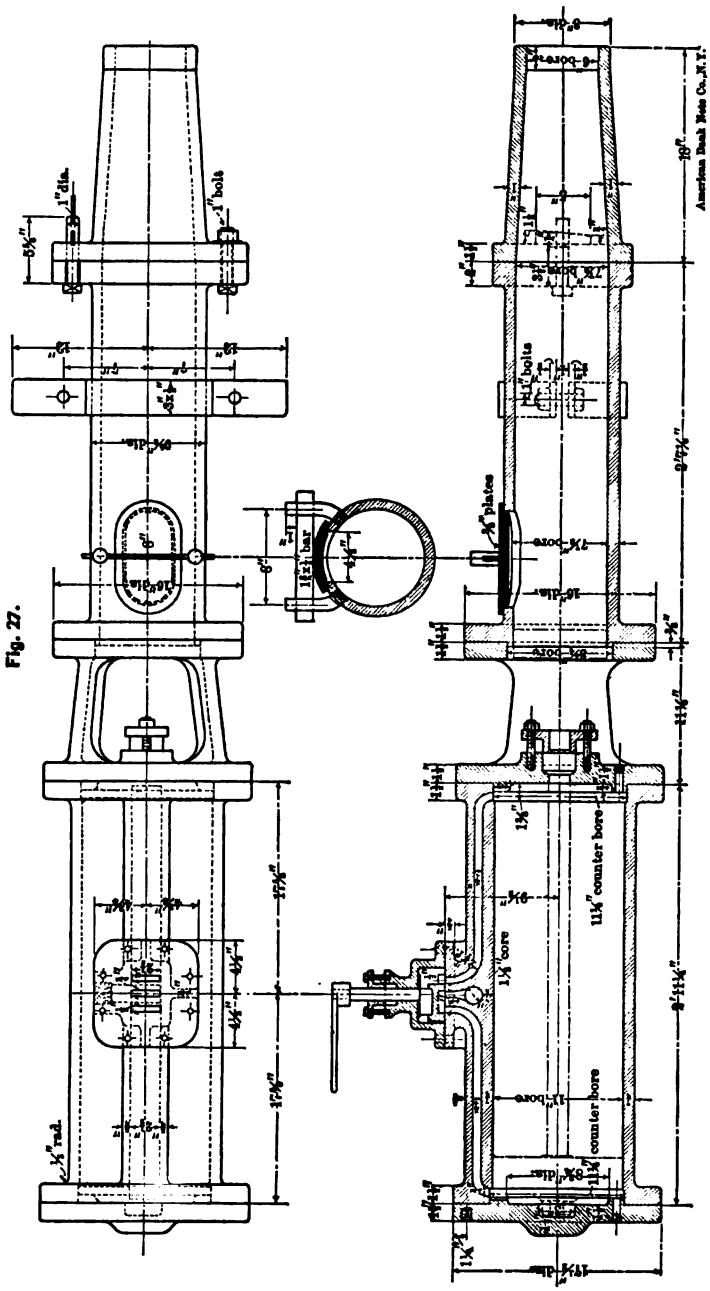


Fig. 27. Vaughen's Tap-Hole Closing-Machine.

staves. The latter are said to be more durable. The repairs to these cars are very light, and they are so easily handled and

cleaned that the problem of economically handling the blast-furnace cinder on the dump may be considered as solved.

The Howden slag-conveyer is a successful machine designed for handling furnace-slag destined for road-ballast. It is an English invention, operated in this country under patents controlled by the Cambria Iron Company, of Johnstown, Pa. The conveyers are manufactured by Messrs. Heyl & Patterson, Pittsburgh, Pa., to whom I am indebted for data and illustrations. The function of the machine is to cast the slag automatically into shallow-pans of cast-iron forming an endless belt, and to cool and granulate the slag so cast by water, delivering it on cars in proper shape for road-ballast. The illustrations, Figs. 30 and 31, show a recent form of the Howden conveyer, as installed at furnaces Nos. 1 and 2 of the Cambria Company. Fig. 30 shows the end of the conveyer nearest the furnace, and the end of the cinder-trough. Fig. 31 shows the conveyer as seen from the cast-house.

The conveyer is 100 feet long from end to end of track. The pans are built of iron castings in three sections, bolted to one common shaft, fitted with supporting wheels, on which the pans roll over tracks constructed of heavy channels. The shafts act also as pins in the heavy link-chain which forms the conveyer. The pans overlap one another, so that no space or joint is left open, where the slag can enter and clog the conveyer.

For the first 25 feet from the end of the cinder-trough the carrying-wheels travel over a horizontal track; they are then depressed in passing through a water-trough 25 feet long (see Fig. 31). The water in this trough is kept at such a level that it surrounds the pans without covering them, the object being to cool the slag without causing it to boil. The last 50-foot section of the conveyer-track has an upward pitch, sufficient to deliver the slag-cakes directly into the railroad-cars. While ascending the slope the pans are freely sprinkled with water, which, however, does not cause a boil, as the slag in passing through the water-trough has been cooled sufficiently to form a crust over its surface. The empty pans return under the bottom of the water-trough and are thoroughly cooled before they again reach the mouth of the cinder-trough.

Granulation of cinder by running it through a flow of water

deserves consideration. The granulated slag will find employment as road-ballast, for slag-brick and for cement manufacture.

Mineral wool, or blast-furnace cinder disintegrated by steam, finds a yearly increasing market.

After all is said, the problem of handling material at the blast-furnace can receive no general solution. Conditions vary, and we must select in each case the means and methods most suitable; but, given the three vital necessaries, plenty of steam, plenty of wind and plenty of heat, then on the judicious and progressive solution of this problem depends the efficiency and success of the plant.

The Cement-Materials of Southwest Arkansas.

BY JOHN C. BRANNER, STANFORD UNIVERSITY, CALIFORNIA.

(Chicago Meeting, February, 1897.)

INQUIRIES are frequently made concerning the chalk- and clay-beds of Arkansas, usually with a view to the manufacture of Portland cement. The chalk-deposits were first described by Professor R. T. Hill in the annual report of the *Geological Survey of Arkansas* for 1888, vol. ii., pp. 153–162. Since the publication of that report I have visited localities then unknown, and have accumulated considerable additional information concerning these deposits, all of which is brought together in the present paper. My report upon the clays of the State has never been published; only a few of the more conveniently available deposits are mentioned in the present paper.

THE CHALK-DEPOSITS.

Stratigraphic Position of the Chalk.—The chalk-deposits of Arkansas belong to rocks of Cretaceous age. The Cretaceous rocks are confined to a single definite area in Southwest Arkansas. Beginning on the Ouachita river, a little northeast of Arkadelphia, its southern or upper margin follows a line about five miles southeast of, and approximately parallel to, the St. Louis, Iron Mountain and Southern Railway from Arkadelphia

to the river-bottoms near the mouth of Bois d'Arc creek on Red river, about ten miles southeast of Fulton, and thence westward to a point between Texarkana and Red river.* The lower and northern border runs westward from the same point on the Ouachita river past Hollywood, Clear Spring, Brocktown, Murfreesboro and Atwood to Ultima Thule.

The chalks are not found throughout these Cretaceous rocks, but are, so far as is now known, confined to the Upper Cretaceous. Professor Robert T. Hill, who has made a special study of the Arkansas Cretaceous, divides it into Upper, Middle and Lower Cretaceous, and places all the chalk-beds in the upper division.†

The divisions made by Prof. Hill are based upon palæontologic evidence, which, for present purposes, is of local importance only; for, according to his correlations, the several chalk-deposits do not belong to the same subdivisions of the Upper Cretaceous. If Prof. Hill's subdivisions hold over a considerable area, the chalk-deposits are simply local variations of different beds.

In the main, however, the points at which chalk occurs lie in a belt which is on the general line of strike of the Cretaceous rocks of the State, as will be seen by referring to the accompanying map, Fig. 1.

The entire series of Cretaceous rocks was deposited in a sea, the northern border of which was originally somewhere north of the present northern border of the Cretaceous, probably at or near the base of the Ouachita mountains. They were laid down upon the upturned edges of the rocks of the Lower Coal-Measures. At the close of the Cretaceous the country to the north was elevated, the sea withdrew southward, and erosion began to remove the soft Cretaceous sediments. This erosion carried away all the northern margin of these beds, uncovering the palæozoic rocks on which they were laid down, and carving the land-surface pretty much as we now find it.

The Cretaceous rocks were in places so deeply decomposed

* This approximate location of the upper limit of the Cretaceous is made upon the authority of Professor Gilbert D. Harris, of Cornell University. See his report upon the "Tertiary of Arkansas," pp. 18-20, and map.

† *Ann. Rep. Geol. Surv. of Ark.* for 1888, vol. ii.; "The Neozoic Geology of Southwestern Arkansas," by Robert T. Hill, Little Rock, 1888, p. 188.

that the soil thus formed conceals the nature of the underlying beds. In other places beds of sediments were deposited on top of the Cretaceous ones. Some of these beds were thin, and when they were afterwards cut through by erosion the underlying rocks were again exposed; but in most instances they were so thick as to completely hide the rocks beneath. All these Mesozoic beds have a gentle southeast dip away from their old shore-line, so that those beds which crop out along the watershed north of Washington, at Columbus and Saratoga, are at a great depth below the surface at Hope and Fulton. The streams have in many places cut their way down through some of the Cretaceous rocks, and in their meanderings have produced wide valleys. The remnants of these beds may be seen occasionally in the tops of some of the hills.

The chalk-bed at White Cliffs has been cut through entirely, so that it now stands 130 feet or more above water on Little river. On the Saline river, however, six miles due east, the same chalk-beds are at the level of that stream.

It should be distinctly understood that chalk occurs in Arkansas only within the Cretaceous area above specified. One may occasionally hear of "chalk-rocks" or "cotton-rocks" in other parts of the State, but in no case are they true chalks. In the region north of the Boston mountains some of the magnesian limestones, sometimes called "cotton-rocks," have been taken for chalks. In the same region a siliceous limestone often forms, upon partial decomposition, a spongy, soft, white rock that is occasionally taken for chalk. It is, however, almost pure silica instead of almost pure limestone. Again, within the Cretaceous area southeast of Murfreesboro, in Pike county, there are considerable beds of white, soft kaolin that has been used in the schools as lump-chalk upon blackboards. In that neighborhood this kaolin is commonly known as chalk. At the northern end of Crowley's ridge, in the northeast corner of Greene county, is a place known as Chalk Bluffs, on account of a white clay exposed at the base of the ridge at this place. Instead of chalk, the material is white pottery-clay, as is shown by the analyses given of it in Owen's first report (p. 21). It contains less than 1 per cent. of lime.

Appearance.—The chalk of Southwest Arkansas has been known hitherto as "rotten limestone." This popular name

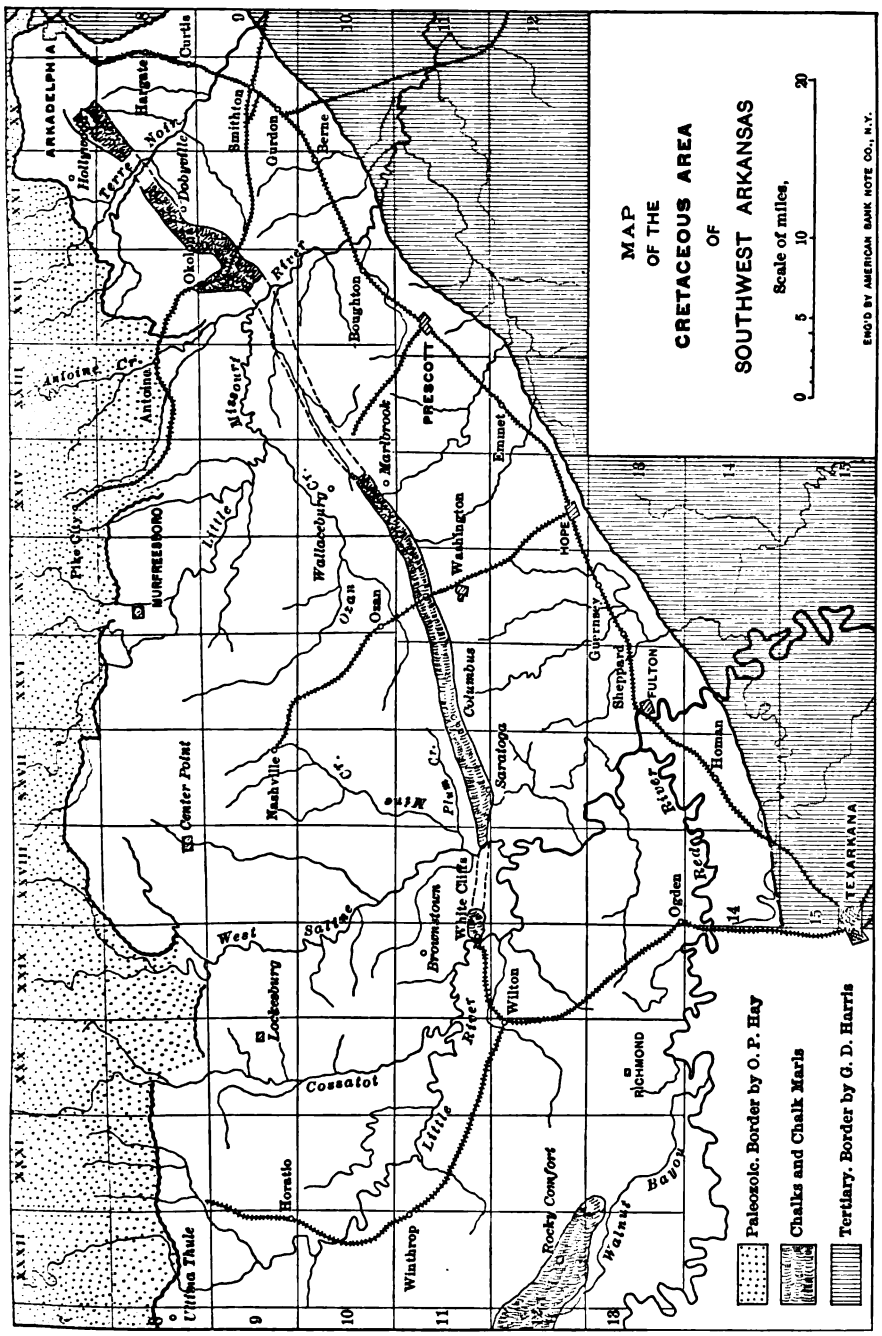

MAP
OF THE
CRETACEOUS AREA
OF
SOUTHWEST ARKANSAS
Scale of miles,
0 5 10 20
ENG'D BY AMERICAN BANK NOTE CO., N.Y.
Paleozoic. Border by O. P. Hay
Chalks and Chalk Marls
Tertiary. Border by G. D. Harris
ARKADELPHIA
Hollywood
Hargate
Curtis
Terre Noir
Dobyville
Smithton
Gurdon
Berne
Okolona
Boughton
PRESCOTT
Antoine Cr.
Antoine
Missouri
Marlbrook
Emmet
Pike City
MURFREESBORO
Little
Wallaceburg
Washington
HOPE
Ozan
Guernsey
Sheppard
FULTON
Columbus
Saratoga
Center Point
Nashville
Mine Cr.
Plum Cr.
Homan
Red River
Ogden
TEXARKANA
White Cliffs
Brownstown
West Saline
Lockesburg
Wilton
Cossatot
Little River
RICHMOND
Rocky Comfort
Walnut Bayou
Horatio
Winthrop
Ultima Thule

FIG. 1.

conveys a general idea of its appearance. Wherever it is uncovered it is soft and earthy in texture, is easily cut with a knife or scratched with the nail, and breaks up in conchoidal fragments when freshly exposed. It is cream-colored to white when dry, or has a grayish cast when damp, and especially when uncovered below the surface of the ground. A microscopic examination of the chalk shows it to be composed principally of the calcareous skeletons of foraminifera. Like most limestones, the chalk forms a rich soil upon decay, and the admixture of organic matter makes this soil very black and sticky. The famous "black lands" and "black waxy lands" of Southwest Arkansas are, indeed, due to the decay of the lime-bearing beds of the Cretaceous area. In some places these soils are in place, resting upon the rocks from which they are made; in other places they have been carried down by water to lower levels and spread out over the "bottoms."

Composition.—According to the books, chalk is "a soft white rock, consisting almost entirely of carbonate of lime in a pulverulent or slightly consolidated state, and readily soiling the fingers when handled." The lime-rocks of the Cretaceous are not all chalks. Many of them are too hard to be classed as such, and, though their composition would not exclude them from available material, their hardness is against them. The softer the rock the more easily crushed, and hence the cheaper.

All the analyses made of the chalks and of the associated rocks are brought together in Table I., (page 58).

Attention should be directed to the fact that these analyses are not all of what are considered good chalks. They were made with a view to determining the nature and availability of the various beds, and they are all given as of some possible value in the efforts of practical cement-makers to settle upon the best rock and to avoid undesirable ones. It will be seen that many of the samples show chalks high in lime and low in deleterious ingredients. The highest percentage of carbonate of magnesia found was 4.01 per cent.; the average is below 2 per cent.

Notes on Leading Deposits.

The following details will be useful to those who may have occasion to locate or use the Arkansas chalks. I shall begin

with the westernmost exposures and follow them eastward as far as they are now known to me.

Rocky Comfort.—The chalk-beds about Rocky Comfort, Little River county, are among the best in the State. At that place they extend over the State-line into the Indian Territory, and eastward they are exposed here and there over the rolling timbered country to and including Section 6 in 13 south, 31 west, a distance of 10 miles. The chalk is not everywhere exposed along this belt, however, for most of the country is covered by gravels, sands and clays, from 3 to 6 feet thick, that conceal the Cretaceous beds. The nature of the concealed chalk-surface is sometimes shown by the character of the well-waters through the region. Some of the waters are hard, as would be expected

FIG. 2.

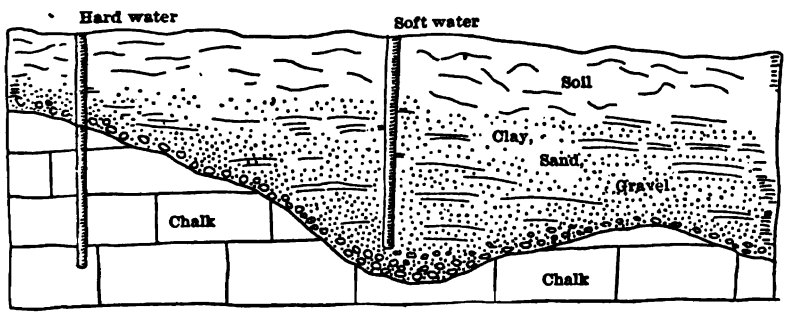


Diagram showing Wells of the Same Depth which Yield Different Kinds of Water.

in a limestone country, but others are quite soft. The reason for this difference is shown by the section given in Fig. 2, which is sufficient explanation of the occurrence of two kinds of water near each other in wells of the same depth.

The hills about Rocky Comfort are capped by gravelly and sandy clayey soil, beneath which, and separating it from the chalk, is generally a bed of coarse pebbles. Three-quarters of a mile north of Rocky Comfort begins a "post-oak flat," the surface of which is a gray sandy clay, streaked with yellow, and containing a few small pebbles. This sandy clay overlaps the lower or northern edge of the entire chalk-area about Rocky Comfort. The chalk beneath these clays is dove-colored, but becomes cream-colored upon weathering. It is usually jointed, and the weathering begins along the joints. A well dug

at the old brick-yard in 12 south, 32 west, Section 8, southwest quarter, passes through the clays and enters the chalk. Chalk was struck also north of Flat creek in Section 6 of the same township and range. Holman's well, in 12 south, 32 west, Section 18, southeast quarter, is 308 feet deep; 10 feet of shaly limestone overlies a blue chalk, which crumbles after long exposure. At 160 feet there are three 6-inch layers of reddish sand. Mr. J. H. Means, who examined the country about Mr. Holman's, thinks the shaly limestone at the top is decomposed chalk. In any case the chalk is more than 300 feet thick at this place, for the bottom was not reached.

A well in 13 south, 31 west, near the center of Section 6, struck the chalk after passing through 6 feet of clay. At Mr. Malden's, on the same section, the chalk was covered by 5 to 6 feet of earth. At a depth of 33 feet the chalk was of a bluish-gray color, and fell to pieces after exposure to the weather.

In 12 south, 33 west, Section 12, northwest quarter of the southwest quarter, a well dug on the low land struck the chalk at a depth of 8 feet.

In the southeast quarter of Section 11 of the same township and range the chalk is exposed in a bluff. In 13 south, 32 west, Section 1, northeast quarter, the chalk is covered by 8 feet of soil and clay. At the center of Section 3, 13 south, 32 west, a well 33 feet deep did not strike the chalk. Wells dug near the eastern boundary of the chalk-belt show that the chalk is covered by 8 to 10 feet of soil and clay, and that the rock is compact and bluish wherever it is penetrated.

At the residence of Col. A. D. Hawkins, east of Rocky Comfort (12 south, 32 west, Section 27), a well 57 feet deep is in chalk for almost the entire distance. A sample of this chalk was examined chemically with the results shown in Table I., Analysis No. 4.

The two samples, Nos. 2 and 3, Table I., were taken from a gully in the southern part of the town of Rocky Comfort; they were about 2 feet apart when in place, and the only difference apparent in them was that of color, No. 2 being cream-colored and No. 3 bluish-gray. The analysis suggests that this difference was partly due to water.

From a personal examination of the chalk-beds about Rocky

FIG. 3.

Chalk Bluffs at White Cliffs Landing.

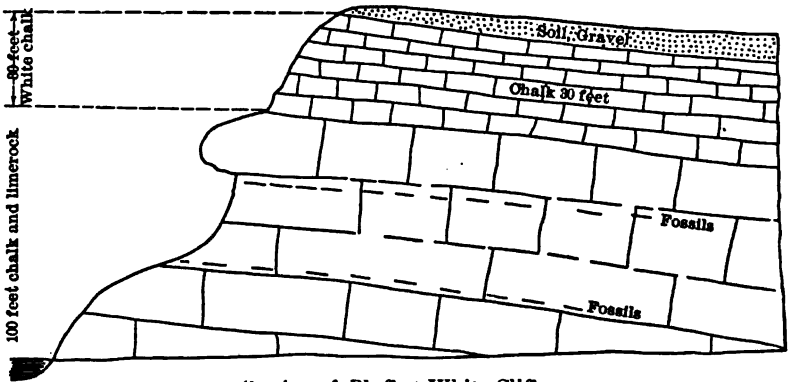
Comfort, I came to the conclusion that there were about 800 acres of the best white-chalk land in that region that could be quarried with little or no stripping of the overlying clays.

White Cliffs.—At White Cliffs landing, on the left bank of Little river, is a notable exposure of chalk (see sketch, Fig. 3). This exposure is in Little River county, 11 south, 29 west, Sections 25 and 36. At this place Little river cuts close to its left bank and leaves exposed a bluff of limestone and chalk about 130 feet high. The chalk proper forms the uppermost bed in the section, and is here about 30 feet thick, with a maximum of 40 feet.

In 11 south, 29 west, Section 35, northeast quarter of the northeast quarter, is the section shown in Fig. 4.

The lower 100 feet contains a good many fossil oysters, and

FIG. 4.



Section of Bluff at White Cliffs.

is not so pure as the 30 feet overlying it. In places the bed immediately beneath the chalk is sandy; this is shown at the base of the cliff at Kinsworthy's old lime-pit, a short distance north of the cliffs. Dr. N. F. Drake, who lately visited White Cliffs, records the following section of the beds beginning at the top of the cliff:

(1) 20 feet massive white chalk. This bed is probably 30 feet to 40 feet thick, but only 15 feet to 20 feet is exposed. This is the bed to be used for cement, etc.

(2) 10 feet of almost white pure chalk, with three bands of more arenaceous rock.

(3) 25 feet of rock, harder, whiter and less arenaceous than No. 4.

(4) 10 feet of rock, harder, whiter and less arenaceous than No. 5.

(5) 20 feet of light-gray arenaceous massive limestone or chalk. *Ostrea vesicularis*(?) common, especially abundant at the top.

(6) 30 feet to the river.

The rocks at White Cliffs landing dip southeast at a low angle, so that the bluffs adjoining the bottom-lands to the south cut off the chalk-exposure in that direction, while the general dip brings them to an end within a short distance northward. That they do not extend northward very far is also shown by the well put down at Col. Coulter's house, about a mile and a half north and a little east of the White Cliffs landing. This well-section is as follows:

Section of Well Bored at D. B. Coulter's, One and One-Half Miles North of White Cliffs.

	Feet.
Waterworn gravel,	3
Clay,	30
Blue lime marl,	200+
White sand, water-bearing,	

Analyses Nos. 13 to 18, inclusive, Table I., show the composition of the White Cliffs beds. Sample No. 14 was taken by Prof. R. T. Hill; the others were collected by myself with a view to getting average specimens, but the material was not systematically sampled.

The White Cliffs locality is six miles in a direct line from the main line of the Kansas City, Pittsburg and Gulf Railway, but a branch line has lately been built from Wilton to White Cliffs. This gives these deposits a considerable advantage they have never before had. Transportation by water is possible during much of the year, but the fact that it could not be depended upon the year round has always stood in the way of the working of the chalks at this place. With the railway in operation the whole year, and the river available a part of the year, the conditions are favorable for extensive development, if railway rates are not prohibitory.

The river-transportation will connect with other railways and markets at Fulton, Ark., at Shreveport, La., and at New Orleans.

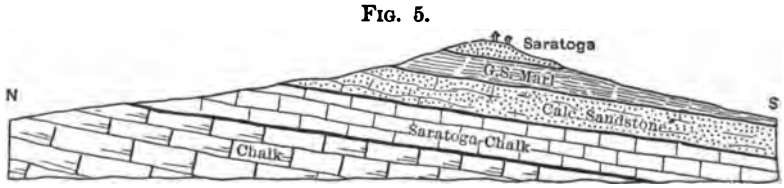
Saline Landing.—Saline landing is six miles due east of White

Cliffs. At the latter place the chalk is on the top of a bluff 100 feet above water, while at the former it is at the water's edge, and the country in the immediate vicinity is low and flat or rolling. I have no doubt, however, that the chalk at White Cliffs and that at Saline landing are the same bed. The southeast dip of the rocks at White Cliffs and erosion will readily account for the difference in elevation at the two points.

Fig. 5, a north-south section through Saratoga (11 south, 27 west, Section 33), shows the general geology of that region. Saratoga is on a hill about 400 feet above the bottom and overlooking the surrounding country.

Saline landing is four miles west of Saratoga on West Saline river, and the chalk-beds shown in the section are exposed at the latter place and thence strike a little north of east, skirting the bottom-lands of Plum creek. East of this they are either concealed by soil or these particular beds thin out and do not appear again in the form of chalk.

FIG. 5.



Section through Saratoga.

At Saline landing the bottoms extend west of the river uninterrupted, so far as is known, to Little river. On the east bank of the stream there is a bank of chalk about 20 feet high exposed for a distance of 300 feet. These beds dip south at an angle of from three to four degrees.

This same chalk is exposed in the deep gullies just east of the stream.

Several years ago three wells were drilled just east of the stream on the farm of which the landing is a part.

The first of these wells passes through the following:

Record of Well Bored at Saline Landing.

Soil and clay,	25 feet.
Brown sand,	1 foot.
White chalk,	140 feet.
Blue marl,	290 "
Sandy bed with pyrites,	10 inches.
Sand in bottom of well.	
	Depth = 456 ft., 10 in.

The following is the record of another well three-quarters of a mile northeast of the landing:

Record of Well Bored Three-quarters of a Mile Northeast of Saline Landing.

Soil,	1 foot.
White chalk,	110 feet.
Blue marl,	277 "
Sandy bed with pyrites,	10 inches.
Sand in bottom.	
	Depth = 388 ft., 10 in.

A thickness of 20 to 30 feet of the chalk is also exposed at this last place.

The first of these well-sections seems to show that the chalk-bed is at least 140 feet thick. The second record gives a smaller thickness of the same bed because it is more eroded at that point which is away from the dip.

Analyses Nos. 10, 11 and 12 show the composition of the Saline landing chalk-beds.

Sample No. 12 was taken from a bed 1½ feet thick, exposed on the creek bank at the landing, that slacks or breaks up more rapidly than the rest of the deposit.

The chalk strikes east and a little north from Saline landing. For the most part it is concealed by the black soil formed by its decay.

It is exposed about 1½ miles northwest of Saratoga, where it forms gently rolling lands. The thickness here is from 30 to 50 feet, and it grades into the overlying marl-beds, which are 210 feet thick. The bottom of the chalk is not exposed at this place. From this point the chalk is exposed eastward to a point about two and one-half miles northwest of Columbus. The bed appears to be of even quality, but it seems to thin out towards the east. Two miles north of Columbus a gully exposes about 5 feet of the chalk. East of this point the chalk-bed of White Cliffs and Saline landing does not seem to be exposed.

Outcrop of the Saratoga Chalk.—The chalk-limestone beds east of Saratoga, west of Columbus, north of Washington and thence eastward to near Arkadelphia are above the upper White Cliffs chalk-bed. But inasmuch as the appearance and composition of these rocks seem to be such as to make them available at

some places for the manufacture of Portland cement, their composition and their general distribution east of Saratoga are here given. The stratigraphic relations of these upper chalks to the lower ones will be seen by referring to the section given in Fig. 5 of the rocks at Saratoga. The White Cliffs beds are shown at the bottom of that section. These are followed above by a series of beds marked in the section as Saratoga chalk, calcareous sandstone, greensand marl and sandstone. The chalk is exposed just west of Saratoga. One mile west of Columbus these beds are 25 to 30 feet thick.

The section in Fig. 5 is not a detailed one. Dr. N. F. Drake gives me the following as the general sequence of the beds above the White Cliffs chalk from Saratoga eastward, as shown by his examination of the geology of the region in question:

(1) Top of the hill at Saratoga; sands.

(2) 100 to 200 feet of sandy black marl.

(3) 20 to 40 feet of rotten limestone.

(4) 10 to 50 feet of bluish yellow marls.

(5) 15 to 20 feet of rotten limestone with *Ostrea vesicularis.*

(6) 200 feet of yellowish marls with *Exogyra ponderosa* at the top.

(7) White Cliffs and Saline landing chalk.

In 11 south, 27 west, Section 35, northeast quarter of the northeast quarter, on the farm of Thos. L. Jones, the rocks of the fossiliferous chalk are exposed. Samples sent from that tract by Mr. Jones were analyzed with the results given in Table I., Nos. 22 and 23. Mr. Jones says some of the rock is of a bluish color, but it bleaches on exposure and finally drops to pieces. The samples analyzed are from near the surface.

Analyses of Fossiliferous Chalks from the Farm of Thos. L. Jones.

	I. Per cent.	II. Per cent.
Loss on ignition, CO_2, etc.,	34.050	40.355
Silica, SiO_2,	16.617	5.002
Iron oxide, Fe_2O_3,	1.512	1.218
Alumina, Al_2O_3,	5.343	2.785
Lime, CaO,	41.370	49.894
Magnesia, MgO,	0.988	0.732
Sulphuric acid, SO_3,	0.000	0.000
Potash, K_2O,	trace	0.000
Soda, Na_2O	trace	trace
	99.880	99.986

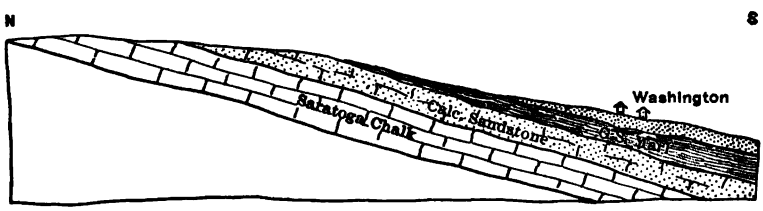
I.—Specimen from gully near surface, 11 south, 27 west, Section 35, northeast quarter of the northeast quarter.

II.—Specimen from block plowed up in field, same section and quarter.

Mr. Jones tells me that a similar rock is found in the northeast quarter of Section 32, in the west half of 27, and in the west half of 26.

The Upper or Saratoga Chalk near Washington.—The outcrop of the upper or Saratoga chalk-bed is not continuously exposed for any great distance; it is for the most part concealed by soil or by an overwash of soil, sands and gravels. The exposures, however, show that it extends from the face of the hill, just east of Saratoga, to Columbus, this latter town standing on or very near its northern margin. From this place it strikes in

FIG. 6.



Section through Washington.

the direction of Washington, but about a mile west of that town it swings northward and crosses the railway on top of the water-shed between Washington and Ozan.

The diagram, Fig. 5, giving the succession of rocks at Saratoga, shows also the succession for Washington and the beds underlying that place. A section at Washington is shown in Fig. 6.

It will be seen that the sections are geologically identical. At Washington, however, the chalk-beds are exposed in the railway-cuts on the water-shed north of and higher than the town, while at Saratoga they are at the base of the hill that slopes away to the north, and are lower than the town.

The Washington exposures examined are in the railway-cuts in 11 south, 25 west, about Section 9, about 2½ miles north and a little west of Washington. Approaching the exposures by the railway-track from the south, the first one shows 7 feet

of light-gray chalk at the top of the cut and 4 feet of leaden-gray beneath, without the bottom of the bed being seen.

The light-gray color of the upper chalk seems to be due largely to weathering; there is, at least, a close conformity of the parting line between the two colors and the contour of the exposed surfaces. The freshly-quarried stone stands with angular faces like a hard rock, but it breaks down rapidly under the influence of the weather. Large, fresh blocks taken from the cut and exposed for one year disintegrated and broke down like so much clay, and the face of the cut within a year came to look more like a bank of clay than a hard rock-face.

Analyses Nos. 6, 8 and 9, Table I., show the composition of average samples taken from the above-mentioned cut.

In the next railway-cut (the one just south of the cut-off made in shortening the railway-track) the rock is less even in texture and has hard spots caused by fossil Cephalopods and Lamellibranchs. A thickness of 10 feet is exposed in this cut. The rock is bluish-gray towards the bottom of the exposure and of a lighter gray at the top. It all disintegrates upon exposure, but not so rapidly as does that in the first cut mentioned above.

No. 7, Table I., is an analysis of the fossiliferous chalk in this railway-cut, made by Dr. A. E. Menke.

The exposures on the railway deserve especial attention on account of their being immediately accessible to railway transportation.

From the point where the chalk crosses the railway north of Washington it runs in a nearly straight line to a point about half-way between Marlbrook and Wallaceburg, where it is cut off or concealed by the overlying sandy beds. Between four and five miles northeast of Washington the chalky bed is about 40 feet thick and seems to be of uniform character. It is capped by about 100 feet of bluish, sandy clay, above which is from 50 to 70 feet of a still more sandy bed. Four miles south-southwest of Wallaceburg it is still forty feet thick, but the bed as a whole varies somewhat. The central part of the bed is somewhat more argillaceous and has more fossils, and the upper four feet have less sand. East of Wallaceburg the upper chalk next appears at and about Okolona. This locality I have not examined personally, but Dr. N. F. Drake, who has visited it for me, says that the chalk similar to that at Saratoga and north

of Washington is exposed in the old fields half a mile south and southwest of Okolona, where it has been eroded so that the total thickness is not to be seen at that place. From two to three miles south of Okolona the full thickness of the bed—20 to 30 feet—may be seen. Here it caps the higher hills. From 3 to 3½ miles south of Okolona the bed dips to the south and disappears beneath gray marls. In this vicinity the material seems to vary in character a little more than usual at different horizons, some beds weathering more rapidly than others. The analyses of samples taken from the beds that break down more rapidly than others upon exposure leave one in doubt about the reason for this peculiarity. In the case of the Saline landing specimens, it was expected that the cause of this disintegration would be readily detected, especially in view of the sharp contrast between the rocks analyzed. In any case this disintegration of the chalk, until otherwise demonstrated, must be looked upon as a valuable peculiarity, as it will render the crushing of the rock less expensive.

A mile west of Dobyville this chalk outcrops along the Okolona-Dobyville road just below a sandy bed. It is about 25 feet thick here, and somewhat harder than usual. The lower 20 feet is quite uniform in character. At Dobyville there is about 100 feet of marl above the chalky bed, the outcrop passing just north of that town. East of Dobyville it is cut off by the bottoms of the Terre Noire. East of these bottoms it follows a broken course past Mt. Bethel church, east of which it has not been located. About five miles west of Arkadelphia, and just below the Rome-Arkadelphia road, where it crosses Big Deciper creek, there is an exposure of 15 to 20 feet of the Saratoga chalk. Here it is quite sandy, and probably not more than one-fourth of the rock is limestone, even in the most calcareous portion of the bed.

Where the road crosses the Little Deciper the top of this bed is at or near the level of the road, and the outcrop lies north of Mr. Haskins's place.

Little Deciper Deposits.—On the south side of Section 23, 7 south, 20 west, where the Arkadelphia-Washington road crosses Little Deciper creek, 4 miles west and a little south of Arkadelphia, the chalky cretaceous beds are exposed on both the east and west sides of the creek. On the west bank these soft

beds are about 25 feet thick, with a few fossil bivalves exposed by weathering on the surface. Above the chalky bed is a bed of brown sandy rock about 6 feet thick, and this is overlain by bright red overwash containing water-worn cobbles. These beds dip gently down the stream.

Table of Analyses of Chalks.

I give below all the quantitative analyses that I have had made of the Arkansas chalks, and also those made by Prof. N. W. Lord, of the Ohio State University, by Dr. Joseph Albrecht, of the United States Mint, New Orleans, and by Dr. Henry Froehling, of Richmond, Va., for the White Cliffs Portland Cement and Chalk Company, and contained in the company's prospectus, published at New Orleans in 1894. I do not know from what part of the beds the samples were taken. So far as I know, these are the only analyses that have thus far been made of them.

Table I.—*Analyses of Arkansas Chalks,*

Giving Silica, Ferric Oxide, Alumina, Magnesia, Lime, Alkalies, Carbon Dioxide, and Water; also the (calculated) Carbonates of Lime and Magnesia.

No.	SiO_2.	Fe_2O_3.	Al_2O_3.	MgO.	CaO.	Alkalies.	CO_2.	Water at 110°–115° C.	$CaCO_3$.	$MgCO_3$.
1	9.77	1.25	*	Trace.	49.55	.....	38.93	.....	88.48	.....
2	12.69	.....	.....	"	46.87	.....	36.83	1.22	83.70	.....
3	12.69	.....	.....	"	45.38	.....	35.66	2.04	81.04	.....
4	10.46	.....	.....	"	45.70	.....	38.28	1.57	86.98	.....
5	8.91	.....	.....	"	49.82	.....	39.15	1.26	88.97	.....
6	24.49	1.68	6.27	0.92	34.63	0.86	29.95	.....	61.84	1.93
7	16.12	1.62	5.64	0.42	42.17	0.41	34.52†	.....	75.30	0.88
8	32.80	1.45	6.32	1.91	25.95	1.66	26.97†	.....	46.35	4.01
9	32.39	1.99	6.63	1.45	25.61	1.50	26.10†	.....	45.73	3.02
10	15.89	1.40	5.46	0.60	42.49	0.35	34.61†	.....	75.87	1.26
11	5.69	1.48	4.22	1.20	48.72	0.26	38.64†	.....	87.01	0.85
12	12.01	1.06	5.29	1.21	44.25	0.32	35.99†	.....	79.01	2.50
13	10.35	1.97	3.02	.....	46.61	.....	36.63	.....	83.24	.....
14	3.49	1.41	*	0.65	52.74	.....	42.16	.....	94.18	1.37
15	4.42	1.03	2.21	.....	53.36	.....	41.93	3.81	95.29	.....
16	5.53	1.34	1.44	.....	51.71	.....	40.63	.....	92.34	.....
17	7.76	1.26	2.58	.....	49.45	.....	38.85	1.15	88.30	.....
18	3.34	0.89	1.80	.....	52.78	.....	41.47	.....	94.24	.....
19	5.04	0.58	3.26	.....	49.92	.....	39.23	.....	89.15	.....
20	6.09	1.20	3.52	.....	49.24	.....	38.69	.....	87.93	.....
21	5.74	0.48	1.02	.....	50.91	.....	40.00	.....	90.91	.....
22	16.62	1.51	5.34	0.99	41.37	.....	34.05†	.....	73.87	.....
23	5.00	1.22	2.79	0.73	49.89	.....	40.35†	.....	89.09	.....

* Alumina included with iron oxide.

† Loss by ignition, comprising carbonic acid and water.

1. Rocky Comfort. 2 and 3. Gully in south part of Rocky Comfort; 2. cream-colored; 3. bluish-gray. 4. Hawkins's well, east of Rocky Comfort. 5. Rocky Comfort, weathered. 6. Upper (7 feet) bed in first railroad-cut, 2½ miles north of Washington. 7. Analysis by Menke of fossiliferous chalk from second railroad-cut north of Washington. 8 and 9. Lower (4 feet) bed from first cut; No. 8 from north end. 10, 11 and 12. Saline Landing; No. 12 a bed exposed on creek which slacks more readily than the others. 13. White Cliffs, base of Kinsworthy's ledge. 14. White Cliffs, sample taken by Prof. R. T. Hill on Col. Coulter's farm, 11 S., 29 W., Sec. 26 (?), exact locality not known. 15. Do., from roadside, N.E. qr. of N.E. qr., Sec. 35, 11 S., 29 W. 16. White Cliffs, light-colored rock below the upper chalk-beds. 17. Do., soft bluish chalk from beneath the chalk-beds. 18. Do., hard bed immediately beneath the chalk-beds. 19, 20 and 21. White Cliffs, precise locality not stated; No. 19 by N. W. Lord; 20 by Joseph Albrecht; 21 by H. Froehling. 22 and 23. Farm of T. L. Jones, 11 S., 27 W., Sec. 35; No. 22 from a gully, near the surface; No. 23 from a block plowed up in the field.

Available Clays.

Clays, to be available for the manufacture of Portland cement, must, besides having the right composition, be sufficiently abundant, uniform in character and convenient to transportation to make them cheap and trustworthy. In no case with which I am acquainted are the surface-clays found in the immediate vicinity of the chalk-deposits to be depended upon. Such clays are, as a rule, too sandy and too thin, and, above all, they are not of uniform composition. Reference is here made especially to the leached sandy clays or "slashes" overlapping the chalk-beds to the north and east of Rocky Comfort, and the clays of the bottom-lands south and west of White Cliffs, and those south, north and west of the chalk-exposures at Saline landing. Fortunately, the Tertiary rocks which overlap the Cretaceous ones to the south and east contain an abundance of excellent clays available for the manufacture of cement. Some of these clay-beds are already being utilized for the manufacture of pottery at Benton and Malvern (Perla switch). At these two points the beds are on the railway. There are many other deposits on and near the railway that are as yet unused, and, indeed, generally unknown. Such occur about Arkadelphia, Malvern, between Malvern and Benton, between Benton and Bryant, at Olsen's switch and at Mabelvale. At Little Rock also there are extensive beds of both clays and clay-shales, while scores of beds of clay-shales may be found along the line of the Little Rock and Fort Smith road to Fort Smith and beyond.

The Tertiary clays at Benton, Bryant, Olsen's switch, Mabelvale and Little Rock are all horizontal or nearly horizontal beds dipping gently toward the southeast. They can be had in many places by stripping off a few feet of post-Tertiary gravel and soil; but in places the covering is too thick to be removed, and the clays can be had only by a system of drifts. In some instances the clay-beds are so nearly uniform in character as to give rise to the idea that they have the same composition throughout. This, however, is true only in a limited sense: the clays do not seem to vary within the small areas thus far worked, but it is highly probable that they will be found to change to sands when a wide area is taken into consideration.

This is not a serious objection to the deposits, however, as there is no possibility of the supply being exhausted.

Cost of Clays.—The lands through the region of Tertiary clays can probably be had for $10 an acre, and for even less than that, as they are generally of small value for agricultural purposes. In some places short switches can be built into the clay-banks, and the clays can be shoveled directly from open cuts into cars. In case it should be necessary to haul to the railway, teams can be had for $2 a day, driver included, and labor at from 75 cents to $1 a day, and outside of the farming season for even less. The employees board themselves and their teams. Hauling must be done in Arkansas mostly in the summer and fall, as the winter and spring rains usually make the common country-roads impassable.

Location of Clays.—The following information regarding the local conditions of occurrence may be useful to those seeking information about the Arkansas clays. Only a few of the many known localities are here mentioned. On account of the geographical relations to the chalk-beds, only those places convenient to railway transportation along the St. Louis, Iron Mountain and Southern Railway southwest of Little Rock are spoken of in this paper. Should a factory, on account of fuel or for other reasons, be located west of Little Rock, clays derived from the Carboniferous clay-shales would have to be used. Of these there is no lack between Little Rock and Fort Smith.

Clays at Little Rock.—There are two general classes of clays

at Little Rock available for cement-manufacture: (1) the *Tertiary clays* that occur in horizontal beds in the southern and southwestern part of the city; and (2) the *Carboniferous clay-shales* exposed in the railway-cuts along the south bank of the Arkansas river, in the cuts west of the town, and in others west of Argenta.

There are other clays about Little Rock and Argenta, such as the chocolate-colored clays along the margins of the river-bottoms, and the pinkish clays forming the high river-terraces and used for bricks on the north side of the river; but these latter two kinds of clays are not available for cement-manufacture, partly because they are too sandy, but also because they are not homogeneous. An analysis of the pink clay of Argenta shows it to contain more than 83 per cent. of silica.

The Tertiary clays were laid down, like the other Tertiary beds of Arkansas, as sediments in water. These strata were originally, and are still, very nearly horizontal. After the old sea-bottom was elevated, however, the ordinary erosion cut down and washed away these soft sediments rapidly, so that now there remains on the hills but a small part of the original beds, and these have been still further obscured by gravels, clays and soils that cover most of the region. The original bedding of these clays is to be seen only in a few deep gullies along the west-sloping hills in the western part of the city, and in the wells that penetrate them. They are of an olive-green to gray color, and upon exposure break up in small cuboidal fragments, or, when thoroughly wet, pack together closely. They are cut in several gullies in the southwestern part of the city, and are penetrated by many of the wells.

The Carboniferous clay-shales are too well exposed in the railway-cut near the upper bridge, and where the electric power-house stands, to require description. Similar shales may be found here and there over a large part of Pulaski county, within the Carboniferous area.

The objection to these shales is that they require grinding before they can be used in cement-making, and the grinding, of course, makes them more expensive.

Composition of the Clays.—The following analyses show the composition of the common run of the pottery clays and fuller's earths of Arkansas. These analyses are of representative samples, and a reasonable assurance may be given that the

clays as found in place are as nearly homogeneous as clays ever are. Most of them contain some sand, usually quite fine. In those cases in which the percentage of sand is given the analyses are of the washed clay. This table might be greatly extended, as I have analyses of a large number of the clays of Arkansas. These, however, will give a correct idea of the general nature of the clays.

TABLE II.—*Analyses of Typical Carboniferous Clay-Shales from Arkansas,*

Giving Silica, Alumina, Ferric Oxide, Lime, Magnesia, Soda, Potash, and Water.

No.	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	Na_2O.	K_2O.	Water.	Total.	Sand in Air-Dried Clay.
1.....	53.30	23.29	9.52	0.36	1.49	2.76	1.36	5.16	97.24	.....
2.....	62.36	25.52	2.16	0.51	0.29	0.66	1.90	5.32	98.72	.....
3.....	58.43	22.50	8.36	0.32	1.14	1.03	2.18	6.87	.....	.....
4.....	65.12	19.05	7.66	0.34	0.31	0.85	1.23	6.12	.....	21.88
5.....	57.12	24.32	8.21	0.72	1.74	0.53	2.07	7.58	.....	.....
6.....	55.36	26.96	5.12	0.30	1.16	1.03	2.69	7.90	100.52	.....
7.....	51.30	24.69	10.57	0.32	0.63	0.72	2.18	9.11	.....	.....
8.....	69.34	22.56	1.41	Trace.	Trace.	2.31	0.04	5.12	.....	.....

1. Clay-shale from railroad-cut at south end of upper bridge, Little Rock. 2. Decayed shale from Iron Mountain Railroad-cut, at crossing of Mt. Ida road, Little Rock. 3. Clay-shale from Nigger Hill, Fort Smith. 4. From Harding & Boucher's quarry, Fort Smith. 5. Clay-shale from Round Mountain, White county, Sec. 6, 5 N., 10 W. 6. From Clarksville, east of college. 7. From S.E. qr. of S.W. qr., Sec. 31, 10 N., 23 W. 8. From N.W. qr., Sec. 23, 1 N., 13 W.

TABLE III.—*Analyses of Typical Tertiary Clays from Arkansas.*

No.	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	Na_2O.	K_2O.	Water.	Titanic Acid.
1.....	63.07	23.92	1.94	0.23	Trace.	1.08	1.15	7.07	.....
2.....	72.44	18.97	1.59	0.18	Trace.	0.91	1.35	5.39	.....
3.....	69.95	22.34	1.44	Trace.	0.08	1.18	1.28	5.98	.....
4.....	71.09	19.86	1.81	0.11	.....	0.81	1.45	5.67	.....
5.....	65.27	18.75	7.34	0.81	1.26	0.81	1.10	6.88	.....
6.....	64.38	17.29	8.25	1.11	0.80	0.42	1.41	6.95	.....
7.....	63.19	18.76	7.05	0.78	1.68	1.50	0.21	7.57	.....
8.....	64.49	23.86	2.11	0.31	Trace.	1.82	0.11	8.11	.....
9.....	67.90	22.07	1.33	0.05	0.59	0.38	1.15	6.86	.....
10.....	48.34	34.58	1.65	0.81	Trace.	1.26	0.44	12.94	1.56.
11.....	62.34	20.63	3.34	0.17	0.67	0.33	0.73	9.34	1.49.
12.....	68.03	17.19	3.00	0.81	1.00	0.54	1.00	6.31	.....
13.....	63.29	18.19	6.45	0.31	2.44	Trace.	0.56	.....	.....
14.....	76.33	16.04	1.24	By dif. 0.99.....				5.40	.....
15.....	75.99	16.12	1.35	By dif. 1.45.....				.....	.....
16.....	73.24	19.61	1.04	By dif. 0.78.....				.....	.....
17.....	45.28	37.39	1.71	1.83	0.29.....			13.49	.....

1. Benton, Hicks's bed, 2 S., 15 W., Sec. 12. 2. Benton, Rodenbaugh, 2 S., 15 W., Sec. 12. 3. Benton, Herrick & Davis's bank. 4. Benton, Henderson's pit, upper bed. 5. Mabelvale, A. W. Norris's well. 6. Olsen's switch, "fuller's clay." 7. "Fuller's earth," Alexander, 1 S., 13 W, Sec. 8, S.W. qr. of S.E. qr. 8. Benton, Woolsey's clay. 9. Ridgwood, 1 N., 12 W., Sec. 25, S.W. qr. of N.E. qr. 10. Benton, Howe's pottery. 11. Clay from 8 S., 15 W., Sec. 4. 12. Clay from 8 S, 15 W., Sec. 5. 13. Clay from 2 S, 13 W., Sec. 13, S. half. 14. John Foley's, 13 S., 24 W., Sec. 18, N.E. qr. of S.E qr. 15. Climax pottery, 15 S., 28 W., Sec. 5, W. half of S.E. qr. 16. Atchison's, 4 S., 17 W., Sec. 24, N.E. qr. of N.E qr. 17. Kaolin, 1 N., 12 W., Sec. 36, Tarpley's.

According to Professor Jameson*, clays for Portland cement should not contain less than about 60 per cent. of silica in combination.

The Manganese-Deposits of the Department of Panamá, Republic of Colombia.

BY EDUARDO J. CHIBAS, NEW YORK CITY.

(Chicago Meeting, February, 1897.)

Location.—The principal manganese-deposits of the Department of Panamá, Republic of Colombia, are located almost due south of Viento Frio, a native village bordering on the Caribbean Sea, and about 45 miles northeast of Colon (Aspinwall). The mining-claims are the Viento Frio, the Carraño, the Nispero, and the Soledad. Each property is to the south of the preceding one in the order mentioned. The first one is about one mile from the coast, and the most southerly boundary of the Soledad is about five miles in a straight line from the same coast-line. I shall confine my remarks to the four properties mentioned and to La Guaca, west of the Carraño, because these are the only ones where mining has so far been done to more or less extent. Manganese has also been found, however, in the neighborhood of these deposits, and a large area adjoining them has been divided into sections and "denounced" as mining-claims. The relative position of the manganese-deposits and Nombre de Dios, the present terminal of the railroad and shipping-port, as well as Viento Frio and the adjacent coast, can

* *Portland Cement*, by Charles D. Jameson, Iowa City, 1895, p. 17.

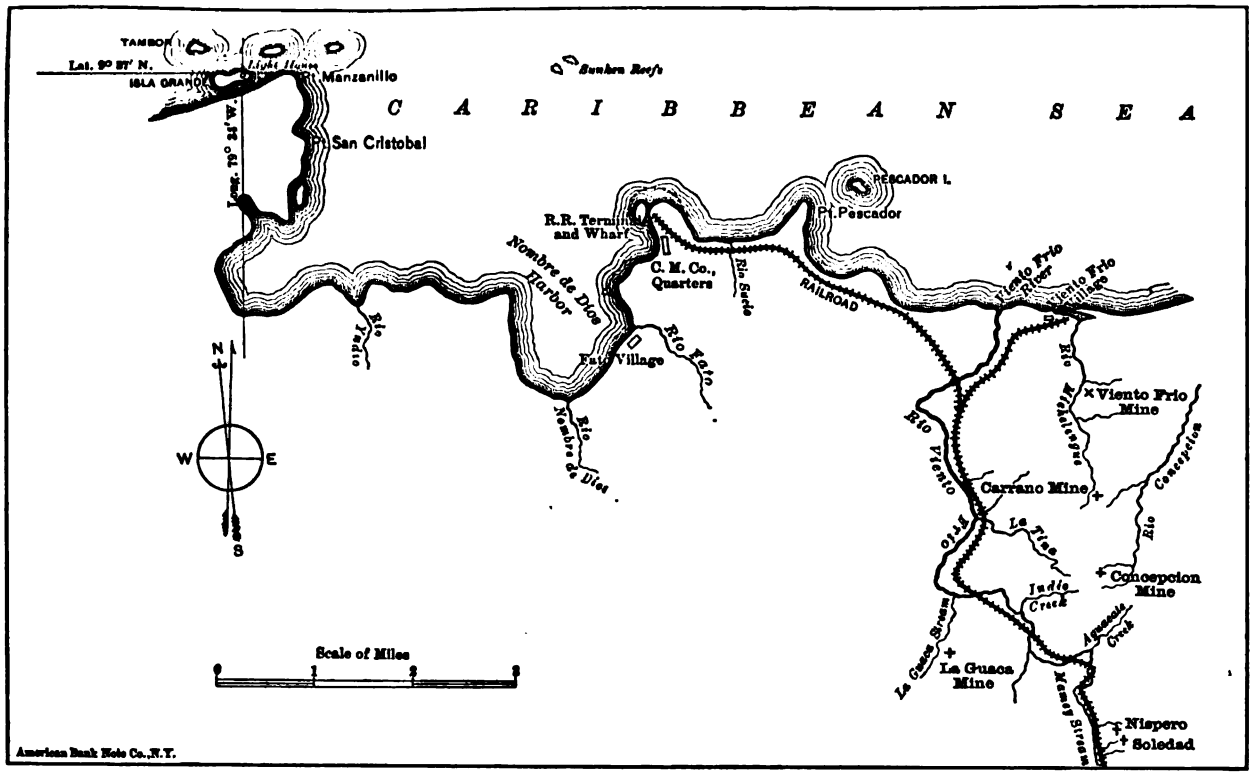
be seen in Fig. 1. The crosses indicate the spots where most of the mining has been done on each claim. The Concepcion mine is within the boundaries of the Carraño claim.

Historical Sketch.—The first shipments of manganese-ore from the Department of Panamá were made between 1872 and 1875, when the Viento Frio mine was worked at irregular intervals and on a very small scale by Mr. Adolfo Steffins. The ore was transported to the coast on mules. The shipments were made to England in small lots of 250 or 300 tons by schooners, which anchored in front of Viento Frio whenever the weather permitted. The ore was taken alongside the ships in canoes or native "dug-outs." I have been unable to get accurate information as to the total amount of manganese shipped at that time; but judging from what I have been able to gather from persons more or less connected with the enterprise, an estimate of 1500 tons would not be far from correct. The only exact data available refer to the last cargo, of 264 tons, which reached England in 1874; it contained, by analysis, 49 per cent. of metallic manganese, and was sold for $18 per ton. Another cargo, shipped from Viento Frio shortly afterwards, never reached England, the schooner being wrecked. This accident precipitated the failure of the enterprise, which had been carried on with very limited means, and in 1875 the mines were practically abandoned.

No further interest was shown in this manganese business until 1890, when Mr. A. M. Potvin, a native of Canada, began prospecting for gold and silver on the Isthmus, and while searching for those metals in the neighborhood of Viento Frio was told that one of his native guides, during a hunting tour, had come across a very rich coal-mine, five or six miles from the coast. Samples of the discovery were secured, and when they were taken to Colon, what was thought to be coal turned out to be manganese-ore. Mr. Potvin then associated himself with Mr. John N. Popham, an American residing on the Isthmus, and Mr. Jos. Piza, an English merchant established in Panamá, for the purpose of placing the newly-discovered deposits in the market.

It is very easy to secure mining-titles in Colombia, especially if, as in this case, the mineral is found on government land. It is then only necessary to denounce as mineral land the tract

FIG. 1.



Sketch-Map, Showing Relative Position of Manganese-Mining Properties, Viento Frio, the Harbor of Nombre de Dios, and the Adjacent Coast.

desired; and after paying a very small fee and complying with some regulations to ascertain that there are no previous claimants, the title is awarded to the petitioner. The law fixes a limit to the extent of each claim, but not to their number. Titles were secured for two adjoining properties comprising the deposits in question, and they were called the Nispero and the Soledad. The latter was named after the "first lady of the land," the wife of the late President Nuñez, and the former after the hill where the manganese was found. The three gentlemen mentioned also secured by purchase the titles to the property that had been previously worked by Mr. Adolfo Steffins. Mr. Popham made several trips to the United States for the purpose of interesting American capitalists in the enterprise, and conducted all the negotiations which finally culminated in the organization of the Caribbean Manganese Company. The property was first brought to the notice of some prominent steel-manufacturers of eastern Pennsylvania; but after an examination, they decided, in view of the difficulties to be overcome, to go no further in the matter. The Carnegies were then approached, and determined to examine these manganese-deposits. With that end in view, a commission, consisting of Mr. H. M. Curry, manager of the Carnegie Steel Co., Mr. W. R. Stirling, then second vice-president of the Illinois Steel Co., and the writer, as engineer, visited the property in November, 1891. Our reports were favorable to the prosecution of further and more thorough investigations, as we considered that the promise of the deposits warranted the expense; but other causes, among which fear of climatic conditions played an important part, prevented the two concerns interested in the investigation from undertaking the work, and the matter was transferred to some Baltimore capitalists. These gentlemen requested the writer to return to Colombia, during the spring of 1892, to make an estimate for the building of a railroad from the Nispero and Soledad mines to the nearest available port, and recommendations for the proper development of the property and for the shipment of ore to the consuming countries.

The writer's report was presented in July of the same year; and shortly afterwards the Caribbean Manganese Company was organized, with headquarters in Baltimore, Md., to carry

out the necessary work to place the ore in the market. Mr. John B. McDonald, of Ryan and McDonald, was elected President of the Company; Mr. John K. Cowen, President of the Baltimore and Ohio Railroad, was made Secretary; and Mr. P. W. Briggs, Treasurer. During the latter part of 1895 they were succeeded by the present officers, Mr. Henry A. Parr, President, and Mr. J. C. Merriken, Secretary and Treasurer. By reason of difficulties, chiefly of a legal nature, inherent to the starting of any new enterprise in a new country, actual work of construction was delayed for some time; but finally in January, 1894, the building of the railroad was begun. In April, 1895, the line from Nombre de Dios to Nispero was sufficiently advanced to allow ore-trains to run over it, and during the succeeding month we inaugurated the shipment of manganese-ore from the port of Nombre de Dios with a cargo of 900 tons. Towards the end of the summer, work was begun on the extension of the railroad from Nispero to Soledad, its present terminus. During the construction of this branch, lack of labor made it at times necessary to stop mining so as to complete the line before the end of the year. The writer was chief engineer and superintendent of the company until the early portion of 1896. He was succeeded in that capacity by Mr. E. G. Williams, the present superintendent. Messrs. R. M. Arango and C. C. Arosemena were Principal Assistant Engineers during the period of construction, the former till January, 1895, and the latter from that date to July of the same year.

Handling and Transportation of the Ore.—As the construction of the railroad and the handling of the ore at the mines have been fully described and illustrated in a paper presented by the writer at the last annual convention, 1896, of the American Society of Civil Engineers,* and in an article in the *Engineering Magazine*, of December, 1896, I will here say only that the railroad from the Soledad to the harbor of Nombre de Dios is nine miles in length, and that portions of the line traverse a very broken and mountainous country, so that it became necessary to adopt maximum curves of 40° (146 feet radius) and maximum grades in favor of traffic of 5 per cent.

* "The Construction of a Light Mountain Railroad in the Republic of Colombia," *Trans. Am. Soc. of Civ. Engs.*, vol. xxxvi., p. 65.

Fig. 2 shows a gravity-incline at the mines, and Fig. 3 an ore-chute between two levels on the hillside.

Character of the Ore.—The mineralogical forms in which the ore occurs, so far as the writer has observed, are braunite, pyrolusite and psilomelane.

The general composition of the ore of the region may be judged from the following analysis of some samples taken from La Guaca mine, without exercising any special care in their selection:

	Per Cent.
Manganese peroxide,	16.98
Manganese sesquioxide,	59.30
Silica,	18.77
Iron oxide,	0.90
Phosphorus,	0.031
Alumina,	0.12
Lime,	0.98
Magnesia,	0.30
Copper oxide,	0.09
Moisture,	0.38
Combined water and loss in ignition,	2.64
Sulphuric acid, ash,	0.10
	100.591

The amount of metallic manganese is 52 per cent. As will be seen from the above, none of the impurities are in sufficient amount to impair the value of the ore for steel-making purposes. The silica is rather high, and a small discount is made in the price of the ore for every unit in excess of 8 per cent.; but by exercising due care in selection, a considerable amount of ore has been shipped containing a much lower average percentage of silica than the above analysis indicates. The amount of phosphorus is insignificant, and less than the allowable limit for a first-class ore.

The analyses by Lehman and Glaser of four sets of picked samples from the Nispero mine gave the following:

	I. Per cent.	II. Per cent.	III. Per cent.	IV. Per cent.
Metallic manganese,	63.74	62.92	62.76	59.98
Iron oxide,	0.28	1.13	0.98	1.19
Silica,	0.25	0.39	5.42	4.71

The amount of phosphorus was below 0.05 per cent. Ores of this class, although they have occurred in small pockets of

uniform purity, are more often found associated with lower grades of ore (higher in silica), and it becomes necessary to exercise great care to prevent the mixture of the two grades. In most of these deposits the highly siliceous ore can be easily recognized at sight, as the silica is visible, especially on fresh fractures, in the form of minute white specks; but when the ore is wet by rain the white specks cannot be seen, and unless a piece is broken, the only means of distinguishing the grade of ore is the weight; the most siliceous being considerably lighter than the least siliceous. Small shades of difference cannot, of course, be distinguished in this manner. Two other causes increased at first the difficulty of selecting the proper classes of ore. One was the inexperience of the miners, none of whom had seen manganese before; the other was the lack at the mines of a chemical laboratory for determining not only the various grades of ore, but also how far they could be combined to make a desirable cargo. This lack made it necessary to wait some time after the cargo had been received in Baltimore before its analysis was known at the mines.

The first cargo of 900 tons analyzed 53 per cent. of metallic manganese and 13 of silica; the next one, of 2560 tons, analyzed 47.5 per cent. of metallic manganese and 20 of silica. Two small cargoes that followed were also high in silica; but when the nature of the ore was better known, and the analyses of several cargoes were at hand, it became easier to set a standard, and several high-grade cargoes have been shipped since. One of 1400 tons, which arrived in Baltimore in February, 1896, is said to have been the richest cargo of manganese-ore ever sold in the United States. The average of the various analyses made by A. S. McCreath and Booth, Garrett & Blair was as follows:

	Per cent.
Metallic manganese,	57.502
Silica,	4.18
Moisture,	2.734
Phosphorus,	0.054

The next cargo, of 1800 tons, was of nearly the same grade. The average of the analyses made by the same chemists, gave: Metallic manganese, 56.788; silica, 5.82; and moisture, 1.415 per cent.

Then followed a cargo of 2600 tons, the average analysis of which, as determined by A. S. McCreath and Lehman and Glaser, was: Metallic manganese, 54.561; silica, 9.88; and moisture, 5.372 per cent.

At present no washing is necessary. The ore is shipped as it comes from the mines. But a large portion of the highly siliceous ore unfit for shipment can, in the future, be rendered merchantable by the establishment of a proper system for crushing, washing and jigging. Such a process was successfully employed at the Crimora manganese mines in Virginia; and if a similar plant, altered to suit the difference in the nature of the ore and local requirements, were established at the Panamá deposits, whenever the amount of that quality of ore would warrant it, it would be conducive to as good results as have previously been obtained at Crimora and at many iron-mines.

Total Exports.—The total exports of manganese-ore from Colombia are estimated as follows:

		Tons.
From 1872 to 1875, estimated at,		1,500
From May, 1895, to December, 1895, by the Caribbean Manganese Co.,	4,550	
During the same period by Brandon, Arias & Philippi, over C. M. Co.'s Railroad,	1,600	
Total for latter half of year 1895,		6,150
From January, 1896, to December, 1896, by Caribbean Manganese Co.,		9,065
Grand total to December, 1896,		16,715

All the ore shipped by the Caribbean Manganese Co. has been from the Nispero and the Soledad, excepting 200 tons from the Viento Frio and Carraño properties. The ore shipped by Brandon, Arias & Philippi was from La Guaca mine.

Topography.—The great mountain-chain of the Andes enters Colombia from the south and divides into three ranges, known respectively as the Eastern, the Central and the Western Cordillera. The Western Cordillera follows a northerly direction, skirting the Pacific coast. As it reaches the Isthmus it describes an arc to the N.W., and extends through the Department of Panamá. The agents of disintegration and erosion have been very active in this Western Cordillera. The originally more or less even slope of the range seems to have been considerably cut up by streams and torrents, which have exca-

vated deep gorges and ravines, and have transformed the mountain-side into a succession of narrow valleys and ridges. On the top and flanks of these lower ridges or hills the manganese-deposits are found.

Geological Features.—The manganese-ore occurs in the form of pockets imbedded in clay, resulting from the decomposition *in situ* of the original enclosing rock. This rock, in places, is only partly decomposed, and its sedimentary origin is revealed, although in some localities metamorphism seems to have turned it to red and yellow jasper. The clay has been eroded in many places, and, leaving some of the pockets uncovered, has given them the appearance of isolated boulders lying on the surface. These boulders vary from a few pounds to 500 tons. In one case, at La Guaca mine, over 1000 tons of ore of good quality were taken out of a single boulder. These boulders have not generally been found of uniform quality. They are often mixed with ore-bearing rock. The clay sometimes assumes various shades of color: red, yellow, white and purple. This is due to the various stages of oxidation and hydration of the manganese and iron it contains.

In some places the removal of the clay that separated several pockets has tended to concentrate and bunch together several boulders which may have been originally many feet apart. Such seems to have been the case in the locality shown in Fig. 4, where a small stream has excavated a narrow gorge on the hill-side, and the carrying away of the clay has brought together boulders that may have been originally separated by a large space filled with the enclosing rock. Fig. 5 shows a pocket of manganese-ore 6 or 8 feet below the surface. The ore is directly in front of the two negroes shown in the picture. The material above their heads is all clay. At this place there was no ore on the ground; but the presence of large stones of jasper stained with black oxide of manganese led us to prospect in that locality, and after going through 6 or 8 feet of clay a pocket with a few hundred tons of ore was found.

Fig. 6 shows one of the manganese-boulders, the upper quarter of which was the only portion above ground. The manganese-ore which had been broken from the boulder before the photograph was taken is piled to the left in the picture.

Near this boulder smaller pockets of ore entirely covered with clay were mined by digging along the side-hill, as shown in the upper left-hand portion of the view.

The rock underlying the manganese-deposits, as far as the writer could observe at the Nispero and Soledad mines, seems to be serpentine. In the neighborhood of the mineral region there are some rocks which to the naked eye appear of igneous origin, but examined under the microscope they have revealed themselves as sedimentary rocks altered by metamorphism. They are, however, composed of fragments not carried very far from their original source, which was an igneous rock, either porphyrite or andesite. The microscopical examinations were made by Prof. J. F. Kemp, of Columbia University. One class of rock he found to consist

"Almost entirely of plagioclase fragments and well-bounded crystals, with which are commingled some small irregular quartzes and a very few patches of chlorite and limonite. The texture is that of a fragmental rock, and we would pronounce it at once a feldspathic sandstone were not the well-bounded plagioclase crystals so striking. I think on the whole it is a fragmental rock, derived, however, from a porphyrite or andesite not far away. The next set of rocks proved to be a much kaolinized felsite or slate—it gives no decisive evidence which—with small amygdules or spherules of calcite. The third variety was much like the previous one, but has enough larger fragments of angular feldspar to indicate a clastic rock, such as a slate."

Prof. Kemp concludes by saying that the slides suggest strongly that the source of the material was from feldspathic rather than quartzose originals.

Most of the rocks in the Department of Panamá seem to have been weathered to considerable depth. In the cuts in the nine miles of railroad, very little unaltered rock was encountered, and the larger portion of the material was so decomposed that it gave no clue to its original nature. Those rocks which had not sufficiently deteriorated to prevent recognition were sandstone, shale and serpentine—all sedimentary or metamorphic rocks and none of igneous origin. No limestones were found in any portion of the line, and no outcrops of that rock have been observed by the writer in that immediate neighborhood.

Geology of Colombia.—Before entering into the discussion of the probable origin and formation of these manganese-deposits, it would be well to give a general abstract of the information

FIG. 2.

Gravity-Incline at the Manganese-Mines, Panamá.

FIG. 3.

Ore-Chute Between Two Levels on Hillside.

FIG. 4.

Manganese-Boulders in Panamá.

FIG. 5.

Pocket of Manganese-Ore in Panamá.

FIG. 6.

Manganese-Boulder Partly Mined, and Neighboring Smaller Pockets.

which I have been able to gather as to the geology of Colombia. Geologists consider that, during the Palæozoic era, all the region now occupied by that Republic was under the sea. Mr. John C. F. Randolph* places the upheaval of the central Cordillera, the oldest of the three mentioned, in the Jurassic period. Towards the close of the epoch of the Cretaceous, Hermann Karsten† states, the two branches adjoining the central Cordillera, that is, the Eastern and the Western Cordilleras, were mostly under water, surrounding, like a longitudinal reef, the firm land of the central Cordillera, with only the highest peaks appearing here and there above water, like a series of elongated islands. The deep arm of the sea that separated and surrounded these islands was gradually being filled with layers of earthy and sandy sediments, brought down by surface-waters and oceanic currents. While the sea was thus growing shallower, the islands were slowly increasing in height and extent; those to the east, the forerunners of the Eastern Cordillera, rising more rapidly than those to the west. In the course of time the reefs connecting these islands were also gradually upheaved and lifted above water, and the Eastern and the Western Cordilleras began to approach more and more their present form, until the end of the Tertiary, when they are supposed to have attained their full height and development. During the upheaval volcanic forces seem to have been very active in some portions of the country, and eruptions of porphyry, granite and syenite also took place. The heat from these igneous masses, as well as the thermal and dynamic effects of upheaval, contributed to the metamorphism of some of the sedimentary deposits. A large portion of the work of erosion and the moulding of the present topography probably took place during the Quaternary; but only a very small portion of land was added to the coast-line during that period.

Origin and Formation of the Manganese-Deposits of Panamá.—The manganese, probably, was originally found as one of the constituents of the igneous rocks of the region, in the form of silicate. Atmospheric agencies, aided by the organic and inorganic acids carried in solution in the surface-waters, caused the disintegration of the igneous rocks; and when the silicate was chemically decomposed, the manganese was carried away

* "Notes on the Republic of Colombia," October, 1889, *Trans.*, xviii., 205.

† *Géologie de l' ancienne Colombie Bolivarienne.*

in solution, until, aided by favorable conditions, it was deposited in the form of oxide or carbonate; but if in the latter state, it was subsequently converted into oxide, as no carbonate of manganese has so far been found in that region. The mineral, as at present found, may have come, not direct from the igneous rocks, but through the medium of sedimentary or metamorphic rocks, with which the manganese may have been deposited, in the manner already stated. When these rocks, in turn, began to decompose, the same process was repeated; that is, the manganese was again dissolved, carried away and redeposited in its present locality.

R. A. F. Penrose, Jr., says:*

"The mineralogical forms in which iron and manganese are deposited from solution, in nature, at ordinary temperatures, depend on the conditions of air and water, whether of an oxidizing or a reducing nature, and on the character of the associated organic and inorganic matter, either in solution or on the floor of the sea, lagoon or bog in which the deposition occurs."

He adds that when solutions of organic or inorganic salts of manganese are freely exposed to the action of air, as in shallow or rapidly-moving streams, or in lakes or some bogs, they are quickly oxidized and may be deposited as more or less hydrous oxide; but an excess of organic matter may cause the formation of carbonate. He also considers that manganese may be deposited sometimes, though very rarely, as sulphide.

In the case of the Panamá deposits, I believe the deposition may have taken place in swamps and lagoons, such as those that at present characterize the topography of portions of the country close to the sea-shore, where the waters draining from the mountain-sides are collected and held for a considerable time before they reach the sea. Sometimes it is only the overflow that finds its way to the sea, along surface-channels, the remaining water disappearing only by seepage and evaporation. At other times the lagoons communicate through a channel with the sea, and become salt- or brackish-water lagoons. I have found no fossils or evidence of any kind to enable me to determine whether the deposition of the ore and enclosing rock took place in fresh or salt water. The waters that carried the manganese in solution carried at the same time argillaceous

* "Chemical Relation of Iron and Manganese," in *Journal of Geology*, vol. i., p. 362. Also *Annual Report of the Geological Survey of Arkansas*, vol. i.

and siliceous particles of the original rock that were deposited and afterwards consolidated into rocks of a shaly nature. Portions of these were afterwards turned into jasper by metamorphism. Later on atmospheric weathering set in, and, decomposing and disintegrating the enclosing rock, turned it into the clay which at present surrounds the ore-bodies. Subsequently, erosion has washed out the clay in many localities and has left the manganese uncovered, in some places, as already described.

The extent and character of the manganese-deposits were probably governed by local oxidizing agencies. Whenever the waters brought down manganese for a continued period, and the local conditions were favorable, large pockets of ore were formed; but when the waters ceased to carry manganese in solution, or the local conditions were unfavorable to its deposition, areas of the associated rock were formed with little or no ore. In some cases the manganese seems to have been deposited with a considerable portion of the sediments carried in suspension, which, disseminated through the ore, give it a highly siliceous character. Such seems to have been the case in some portions of La Tina, where a considerable amount of highly siliceous ore has been found. At this place the bedded character of the deposit is more plainly discernible than in any of the other localities observed by the writer.

It is possible that some of the high-grade ore at present mined at the Nispero and the Soledad may have been the product of secondary chemical changes after the upheaval. That is, when the enclosing rock began to decompose the manganese may have been redissolved and redeposited in the clay below and not far away. Again, in some localities the manganese, when deposited, may have been so finely disseminated through the associated rock as to have been of no economic value; but, later on, by a process of resolution and redeposition, it may have been concentrated into bodies of comparatively pure ore and in workable quantities.

As to the age of the deposits, I can only say that no clue has been found to determine at what period they were formed; but if my theory as to their origin is correct, and they were formed just as that portion of the country was emerging from the sea, they probably belong to the Tertiary horizon, although the subsequent secondary chemical changes may have taken place at a more recent period.

It is with reluctance that I have advanced an opinion as to the origin and formation of these deposits, as I do not consider the data I have been able to gather from personal observations sufficient for a thorough treatment of the subject; and I freely confess that future and more extended investigations may serve to modify my conclusions materially.

Sorting Before Sizing.

BY ROBERT H. RICHARDS, BOSTON, MASS.

(Chicago Meeting, February, 1897.)

THE adaptation of European methods of concentrating ores to suit the conditions of this country has followed the lines that simplify machinery, diminish labor and increase capacity. Noteworthy instances are the substitution of hydraulic classifiers for the last one or two trommels in the series, and the substitution of a distributing-tank for the slimes in the place of the *Spitzkasten*.

The question may be fairly asked, At what point do losses become serious in consequence of these changes? It is to study this question that the investigation herein described has been undertaken. After the losses are located and the methods of stopping them are pointed out, it will then lie with the individual mill to decide whether or not the plans proposed will save sufficient value to warrant their adoption.

In the discussion of sorting followed by sizing, as the terms are used in this paper, we have to deal with the preparation of slimes by the *Spitzkasten* for subsequent washing upon the slime-table.

The *Spitzkasten*, or pointed box, is an inverted hollow pyramid, generally of wood, usually with four faces or sides coming together in a point below. The sides can have, according to Rittinger, a minimum angle of slope of 50° with the horizontal. The feed-water carrying fine slimes (less than 0.01 inch, or 0.25 mm. in diameter) for treatment is fed at one end and carefully distributed over the whole width. The outlet is over the opposite end. The water-current passes direct from inlet to outlet, and has usually been regarded as a surface-layer pass-

ing over without much disturbing the stagnant layer of water below. Under such conditions the coarser grains of quartz, together with finer grains of the heavier metallic mineral—for example, galena—drop out of the current into the stagnant water below, and, settling to the apex, are discharged through a pipe or spigot. Several pointed boxes in series, each larger than its predecessor, will give a series of products, each of which is finer than its predecessor, and the particles will be approximately sorted according to the law of free-settling particles. This process of grouping together grains of metallic or specifically heavier mineral, which are smaller in size, with those of the non-metallic or specifically lighter mineral that are relatively larger in size, is called *sorting* the slimes, and the series of products are called *slime-sorts*.

A slime-table for washing ores has a true surface, either a plane or a much-flattened cone, with gentle slope, and upon the upper margin of it water is evenly distributed. The layer or film of water has an approximately even thickness if the table is plane, or a gradually decreasing thickness if the table is a convex cone, varied only by a series of slight waves, and is continuously moving down in the direction of steepest slope. If, now, one of the above sorted products is placed upon such a slime-table, the particles of galena, for example, being small, settle upon the table and do not move, or move but slowly because they are in the lower slow-moving portion of the water-film, while the quartz-grains, being larger, reach up into the quick-moving water and are carried forward rapidly.

The operation of separating small grains from large grains is technically called *sizing*, whether it is done by putting the grains on a sieve of such a mesh that it allows the smaller grains to pass through while it holds back the larger grains, or, on the other hand, is done by a slime-table, which rolls the large grains rapidly towards its margin, but moves the small grains slowly or not at all.

By combining the two principles—namely, the *Spitzkasten* and the *slime-table*—we have what is called separation by *sorting followed by sizing*.

There is a dearth of exact data upon the laws defining the water-quantity and slope-angle best adapted for the treatment of the different slime-sorts upon slime-tables, and it is to supply

this deficiency that the present investigation has been undertaken. The different writers give us figures derived empirically from mill-practice.

Linkenbach gives for stationary slime-tables:

	Slope of slime table. Degrees.		Per cent.
For 1st spigot-product, . . .	6° 20′	or	11.11
" 2d " . . .	5° 43′	"	10.
" 3d " . . .	4° 46′	"	8.33

and for revolving slime-tables:

	Slope of slime table. Degrees.		Per cent.
For 1st spigot-product, . . .	5° 43′	or	10.
" 2d " . . .	5° 12′	"	9.09
" 3d " . . .	4° 46′	"	8.33

Kunhardt gives as

	Slope of slime table. Degrees.	Per cent.
European practice, ...	5° to 10°	or 8.75 to 17.63

United States practice is illustrated by the list on the following page, comprising 25 slime-tables, in 20 mills, located in about 10 different districts. In this list *a* represents convex conical revolving tables; *b*, convex stationary tables, and *c*, concave conical revolving tables.

To design machines of this class intelligently, we need to have exact data as to the angle at which:

1. The quartz starts to roll.
2. The quartz is all moving.
3. The heavy mineral starts.
4. The heavy mineral is all moving.

These observations should be taken with many different slime-sorts and quantities of water, ranging from very small to very large, to cover all possible demands.

To obtain the desired experimental data, the author required at the outset a set of products as perfectly sorted as possible. For this purpose the sorting-tube, Fig. 1, was designed. Hydraulic water is fed at *e* at a constant rate, admitted by a dial-cock at constant pressure, guaranteed by an overflow column-pipe to give constant head. This passes up and overflows at *i*, at any desired speed. If *k* be the average area in square centi-

meters of the tube cd, then the grammes or cubic centimeters l of water delivered per minute at i, divided by k, will give the upward velocity m of the water in centimeters per minute:

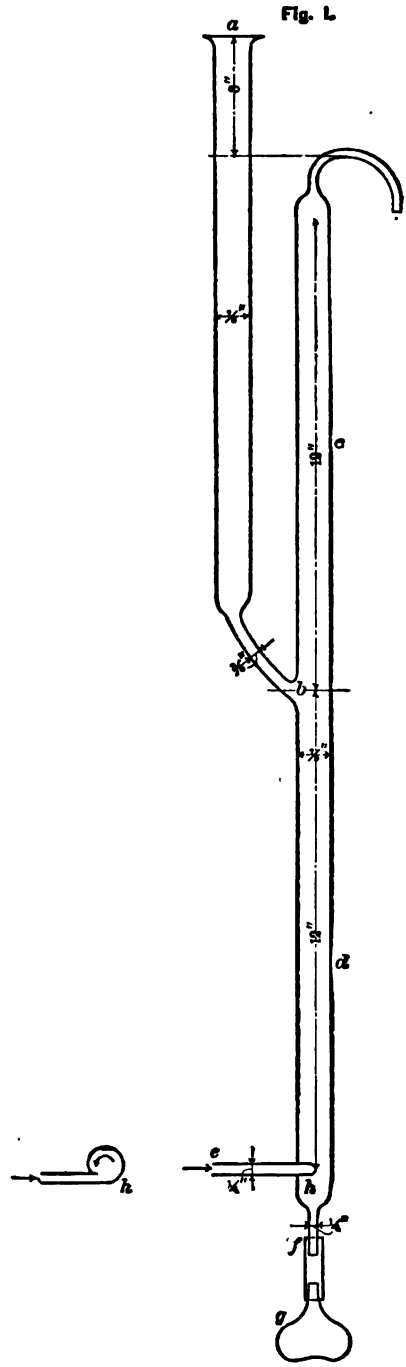
$$m = \frac{l}{k}.$$

The same result may be obtained in inches per minute by weighing the water in pounds as follows: The pounds of water

List Showing Angles of Slope in American Practice.

Name of Mill.	Kind of Table.	Material.		Slope.	
		Concentrates.	Waste.	Inches per Foot.	Degrees.
Rocky Mountain....	*a.*	Pyrite, Chalcopyrite.	Quartz.	1.	4°47′
Silver Age.....	*b.*	Galena, Pyrite.	Quartz, Feldspar.	0.94	4°30′
Kohinoor.....	*a.*	Galena, Blende.	Flint.	1.05	5°
Central Lead Co.....	*a.*	Galena, Pyrite.	Limestone.	1.05	5°
Flat River Lead Co..	*a.*	Galena, Pyrite.	Dolomite.	1.25	6°
Mine la Motte.....	*a.*	Galena, Pyrite.	Limestone.	1.35	6°25′
Revenue Tunnel..	*c.*	Galena, Gray copper.	Quartz.	1.33	6°20′
	a.			1.24	5°55′
	b.			0.72	3°25′
Smuggler.....	*c.*	Galena, Pyrite.	Quartz, Blende, Limestone.	1.74	8°15′
	a.			1.37	6°30′
	c.			1.58	7°30′
	a.			1.05	5°
Bunker Hill and Sullivan.....	*a.*	Galena, Pyrite.	Siderite, Quartzite.	1.25	6°
Gem.....	*a.*	Galena.	Blende, Quartz.	1.125	5°21′
Helena and Frisco..	*a.*	Lead sulphide, carbonate and phosphate.	Quartz.	1.125	5°21′
				1.25	6°
Last Chance.....	*a.*	Galena.	Quartz.	0.96	4°35′
				0.75	3°35′
Standard.....	*a.*	Pyrite, Galena.	Quartz.	1.25	6°
				1.50	7°8′
Stem Winder.....	*a.*	Galena.	Quartz.	1.00	4°47′
Anaconda.....	*a.*	Copper sulphides.	Quartz and Feldspar.	0.75	3°35′
Butte Reduction.....	*q.*	Copper sulphides.	Quartz and Feldspar.	1.125	5°21′
				1.000	4°47′
Calumet and Hecla..	*a.*	Native copper, Martite.	Rhyolite.	1.54	7°25′
Osceola.....	*a.*	Copper, Magnetite.	Calcite, Amygdaloid.	1.25	6°
Quincy.....	*a.*	Copper.	Amygdaloid.	1.25	6°
Tamarack.....	*a.*	Copper.	Rhyolite.	1.25	6°

Fig. L.



Sorting-Tube.

at 62° F., p, delivered per minute at i, multiplied by 27.712, gives cubic inches, and this product divided by the area r in square inches gives the ascending velocity s in inches per minute:

$$s = \frac{27.712\ p}{r}.$$

The sorted products were obtained by feeding at a, in very small quantities at a time, mixed grains of quartz and galena, which had passed through a limiting-sieve of 10 meshes to the linear inch, and therefore contained grains of both minerals ranging from that size down to dust. The grains became subject to the action of the current at b. If they were light enough to rise in the current flowing at any given time they were discharged at i. If heavy enough to fall, they passed down to the bulb g. The rising portion c of the tube is 12 inches long, to give a heavy grain that distance to repent and return. The falling portion d of the tube is also 12 inches long, to give a light grain the same opportunity. A rotary motion is given to the water in d, to prevent a downward current on one side and an excess of upward current on the other. If the mixed quartz and galena is fed slowly, so that "free-settling" conditions prevail* while the water is rising—for example, 40 millimeters per second—it yields two products: the overflow-grains which rise in a 40-mm. current, the bulb-grains which fall in the same current. If, now, we feed the above bulb-grains to the tube when the water is rising 50 mm. per second, it again yields two products: overflow-grains which rise in a 50-mm. current, and bulb-grains which fall in the same current. This second overflow is defined as consisting of grains of quartz and galena which, under free-settling conditions, fall in a 40-mm. current and rise in a 50-mm. current.

To obtain a complete set of products, as perfectly defined as the above, the water-current was rated at frequent intervals all the way from 0.0496 inches (1.261 mm.) per second to 7.8259 inches (198.777 mm.) per second. And to obtain further information, the length and width of each of ten grains of quartz and also of galena were measured by microscope-micrometer for each product. The average of the ten measures of

* See "Cycle of the Plunger-Jig," *Trans.*, xxvi., 3.

length and of width (twenty in all) is called the average diameter of the grains. These figures are given in inches and in metric measures in Table I., together with the ratio of the average diameters of quartz and galena. The specific gravity of the quartz is 2.640 and that of the galena is 7.586.

TABLE I.—*Diameters of the Quartz and Galena Particles Which are Equal-Settling in the Upward Currents Specified when Treated Under Free-Settling Conditions, together with the Observed and Calculated Diameter-Ratios.*

Particles Fall in Current of Inches per Second.	Particles Rise in Current of Inches per Second.	Diameter of Particles in Inches.		Ratio Between Diameters of Particles Actually Obtained.	Diameter of Particles in Millimeters.		Particles Fall in Currents of Millimeters per Second.	Particles Rise in Currents of Millimeters per Second.	Ratio Between Diameters of Particles Averaged by a Curve.
		Quartz.	Galena.		Quartz.	Galena.			
.000	.050	.00119*	.00076*	1.55	.0301*	.0194*	0.00	1.26	1.54
.050	.099	.00132	.00078	1.69	.0335	.0198	1.26	2.51	1 68
.099	.199	.00224	.00115	1.95	.0568	.0292	2.51	5.05	1.82
.199	.292	.00304	.00162	1.87	.0772	.0412	5.05	7.42	1.96
.292	.394	.00387	.00192	2.01	.0982	.0488	7.42	10.01	2 09
.394	.577	.00561	.00242	2.32	.1423	.0613	10.01	14.68	2.23
.577	.780	.00739	.00284	2.60	.1875	.0721	14.68	19.80	2.35
.780	1.186	.0089	.0041	2.18	.2254	.1032	19.80	30.12	2.48
1.186	1.589	.0135	.0051	2.62	.3416	.1305	30.12	40.37	2.61
1.589	1.972	.0153	.0055	2.76	.3880	.1404	40.37	50.08	2.72
1.972	2.366	.0206	.0067	3.07	.5241	.1708	50.08	60.09	2.82
2.366	2.769	.0232	.0079	2.95	.5892	.1997	60.09	70.34	2.92
2.769	3.160	.0260	.0094	2.77	.6590	.2381	70.34	80.28	3.03
3.160	3.552	.0339	.0108	3.13	.8604	.2750	80.28	90.21	3.12
3.552	3.919	.0403	.0135	2.99	1.0234	.3428	90.21	99.54	3.21
3.919	4.334	.0450	.0138	3.26	1.1424	.3504	99.54	110.09	3.29
4.334	4.726	.0521	.0144	3.62	1.3216	.3648	110.09	120.03	3.36
4.726	5.135	.0560	.0149	3.77	1.4224	.3776	120.03	130.43	3.42
5.135	5.526	.0562	.0166	3.39	1.4256	.4208	130.43	140.37	3.49
5.526	5.918	.0632	.0180	3.52	1.6032	.4560	140.37	150.31	3.54
5.918	6.303	.0664	.0181	3.67	1.6848	.4592	150.31	160.09	3.59
6.303	6.691	.0689	.0182	3.78	1.7488	.4624	160.09	169.95	3.63
6.691	7.106	.0710	.0207	3.44	1.8032	.5248	169.95	180.51	3.66
7.106	7.826	.0778	.0228	3.42	1.9744	.5776	180.51	198.78	3.70

Before passing to the next step in our subject it is interesting to note that in the sorting by the sorting-tube, Fig. 1, the values here obtained for grains settling under free-settling conditions corroborate those given in "Close Sizing Before Jigging,"† showing again that for fine sizes the free-settling ratio

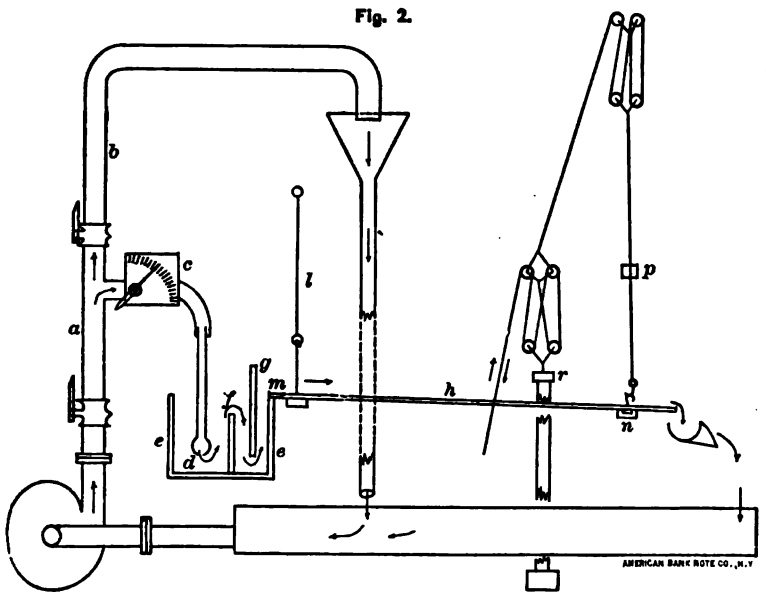
* These averages have less value than the others, because the diameters in this case range from the figures given down to zero.

† *Trans.*, xxiv., 409.

of the diameter of the quartz particle to that of the galena particle is less than Rittinger's formula would give, and that the ratio decreases as the diameters decrease (see Table I.). The measures of the diameters given in "Close Sizing Before Jigging" were made by sieves. The measures for this paper were made by micrometer in a microscope, and are therefore probably more accurate than those of the former.

The sorting apparatus having been obtained, and with it a very perfect set of sorted products, it was necessary next to

Fig. 2.



Laboratory Slime-Table; Elevation.

prepare an accurate sizing-apparatus or slime-table. This is shown in elevation in Fig. 2 and in plan in Fig. 3.

This apparatus (see Fig. 2) consists of a plane plate-glass *h*, with ground surface, capable of being fed with an even flow of water and of being tilted at any angle. The details are as follows:

a. A hydrant of 2-inch pipe supplied with water by a centrifugal pump.

b. A column-pipe, 8 feet high, to completely even up the pressure.

c. A dial-cock and hose for delivering constant quantities of water to the table.

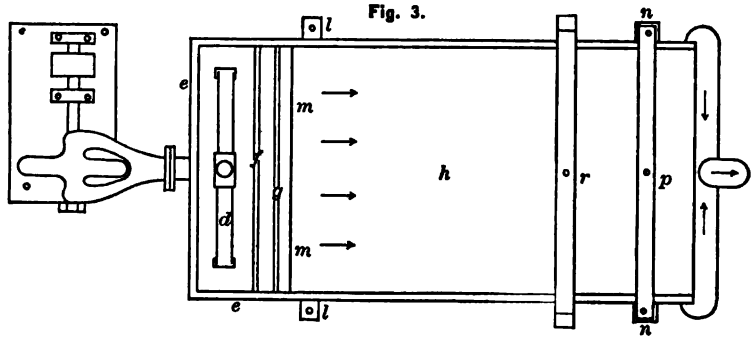
d. A long rose-pipe delivering water the whole length of the distributing-tank *e*, which has a dam *f*, a baffle-plate or gate *g*, and an outlet over a straight edge, *m*.

h. A plate-glass ground surface, 2 feet wide, 4 feet long, suspended on two rods, *l*, at the head, and at the foot by a block and tackle, multiplying the movement 20 times by 4 double pulleys.

This table has a surface well adapted to its work. It appeared to be rough enough to cause the particle to roll, but not rough enough to obstruct its passage.

It can be tilted at any angle from level to 45° slope and adjusted with minute accuracy. The angles were measured

Fig. 3.

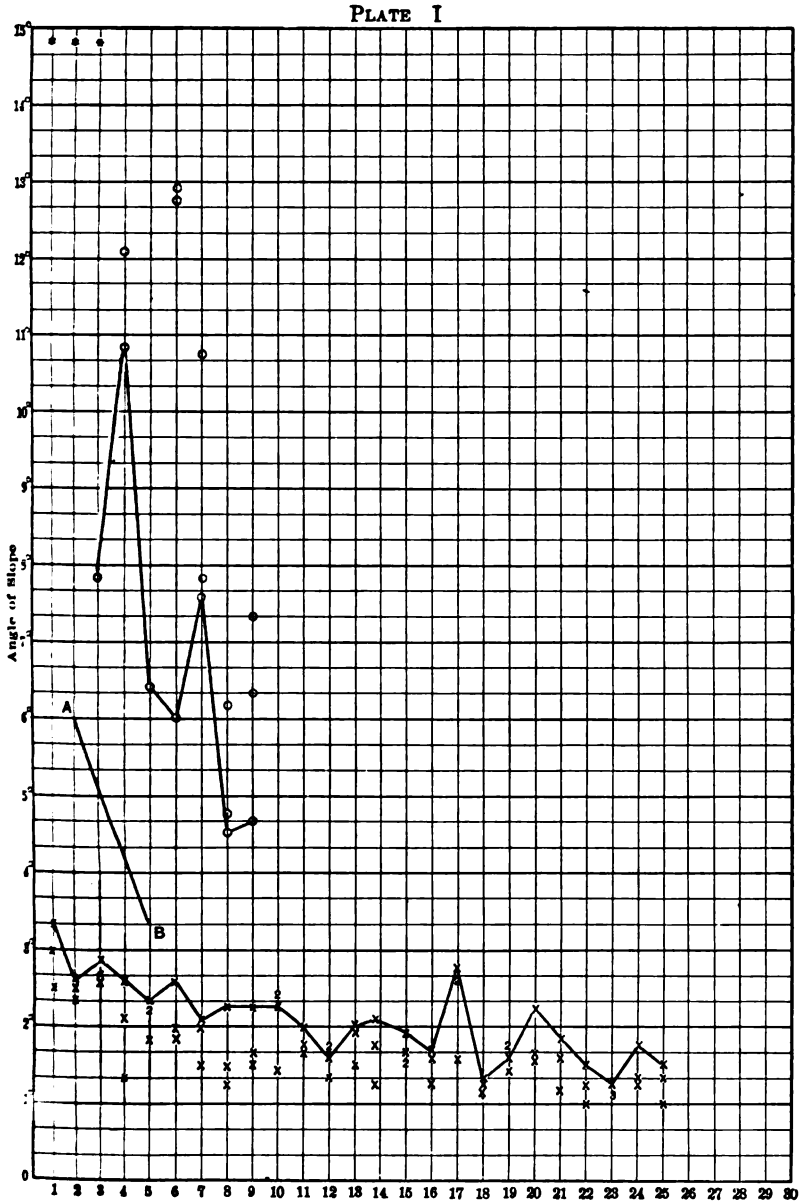


Laboratory Slime-Table; Plan.

by a Linton level, with a probable error of five minutes of arc; the film thickness on the crest and trough of the waves was measured by a micrometer gauge reading to 0.001 inch, with probable error in reading of 0.002 inch on wave-crests and 0.004 inch in the troughs, and where there was no wave, 0.002 inch. The sands to be treated were put on at 23 inches from the head and spread out in a straight line about 8 inches long at right angles to the direction of flow to prevent the grains from hindering each other. The quantity of water flowing was weighed in a bucket with a spring-balance weighing to sixty pounds, and reading to one-tenth of a pound, the limit of error being about 1 per cent.

An experiment is begun by gauging the water to a given number of pounds flowing off the tail. The charge of grains of quartz and galena is then put in place and the table is slowly tilted until (1) the first grain of quartz starts. Then,

PLATE I



* The angles of galena, except the one plotted, were all above 20°.

the angle of slope and the thickness of film on crest and trough are taken. The slope is increased gradually till each of the following events has occurred, stopping to take the above measures at each point:

(2) All the grains of quartz move.
(3) The first grain of galena moves.
(4) All the grains of galena move.

After these four sets of observations are concluded, the water is set at another rate of flow in pounds per minute and the whole series is recorded again. In this way the conditions are studied with small intervals, from a quantity of water which breaks down because it is too little at one end of the series, to a quantity which is absurdly large at the other end. The little end of the series for small grains was where the water failed to cover the table, for large grains, where events 2 and 4 occurred at nearly the same angle.

It will be observed that the method employed was to measure angles, films, etc., at the moment when a body at rest begins to move. If the opposite plan, namely, to measure angles, films, etc., when a moving body comes to rest, had been adopted, it would have represented more exactly what happens upon a convex conical slime-table. The difficulties of making the tests by the latter method caused the selection of the former. The difference in results is that the figures for angles given in the table are slightly larger than they would be if the coming-to-rest method had been employed.

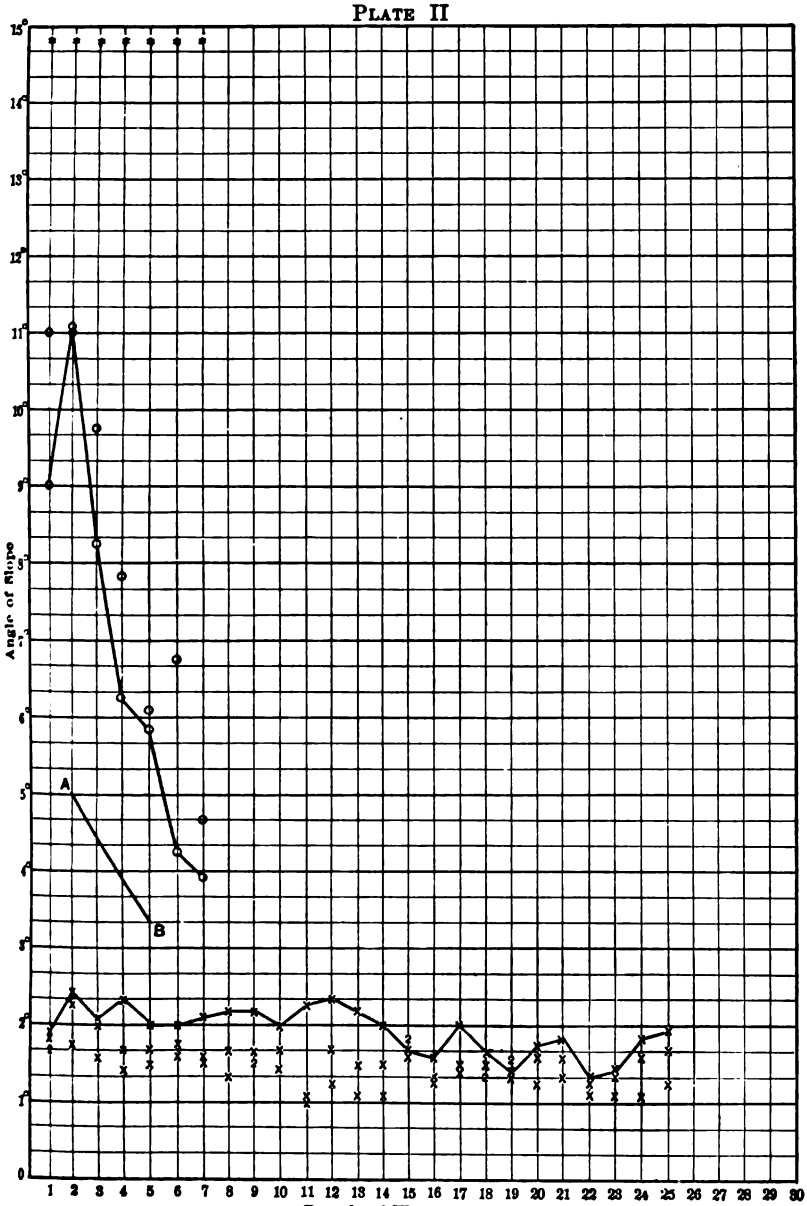
A portion of the measurements of angles obtained in this work are presented in Tables III. and IV.

Table III. gives the slope-angles at which all the particles of each of the different sorts of galena start to move. In most cases each figure is the lowest of three observations. They are called the finish-angles of galena.

Table IV. gives the slope-angles at which all the particles of each of the different sorts of quartz start to move. In most cases each figure is the highest of three observations. They are called the finish-angles of quartz.

This finish-angle of quartz is believed by the author to be the most important angle of all. From a table set at this angle nearly all the quartz will roll off, and nearly all the galena will remain on it at rest.

PLATE II



Grains which fall in an upward current of 2.51 mm. per second and rise " " " " " 5.05 " " "

o – Galena of .0292 mm. diameter. x – Quartz of .0568 mm. diameter.

* The third angle of galena was above 20°.

The starting-angles did not appear to the author to have the same value as the finish-angles in guiding the judgment for best slope and water for a given slime-sort, and have, therefore, been omitted.

In examining the starting-angles of galena and quartz it was noticed that the first grain of galena often starts at an angle smaller than the finish-angle of quartz. This is not as serious a matter as it appears, for several reasons. First, there were only a few light grains of galena that started before the quartz; secondly, they moved slowly, so that the quartz easily overtook and left them behind; and, thirdly, the occurrence was confined to those experiments that were near the least water-quantity, and therefore outside of the range of the best working conditions.

A few representative examples of the finish-angles of quartz and galena are plotted in Plates I. to VI. The vertical dimension (ordinate) gives the angle of slope; the horizontal dimension (abscissa) gives the amount of water per minute for 2 feet of width. The sizes of the grains and the sorting currents that are selected for plotting are given in Table II.:

TABLE II.

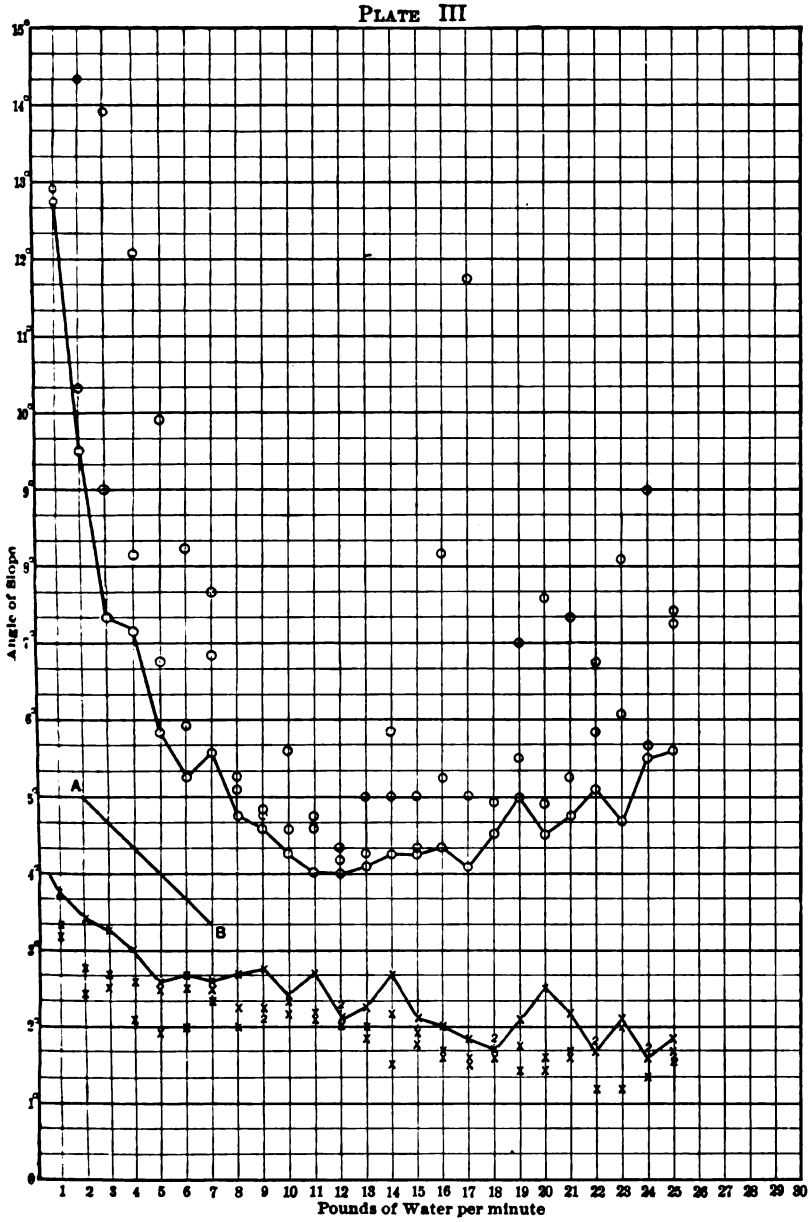
	Millimeters per Second Upward Current.		Average Diameters.		Ratio.
	Grains Fall.	Grains Rise.	Quartz.	Galena.	
	Millimeters.	Millimeters.			
Plate I	1.26	2.51	.0335	.0198	1.69
Plate II	2.51	5.05	.0568	.0292	1.95
Plate III	10.01	14.68	.1423	.0613	2.32
Plate IV	30.12	40.37	.3416	.1305	2.62
Plate V	40.37	50.08	.3880	.1404	2.76
Plate VI	60.09	70.34	.5892	.1997	2.95

Three experiments were tried upon each water-quantity for each slime-sort, and the three readings are given upon the plates.

To aid in interpreting the curves, a line has been drawn joining the largest quartz-angles, and also the smallest galena-angles for each water-quantity.

The finish-angles of galena have been erased from Plates I. and II. for values of water above 8 pounds, since great difficulty was

PLATE III



Grains which fall in an upward current of 10.01 mm. per second and rise " " " " " 14.68 " " "

o Galena of .0613 mm. diameter x Quarts of .1453 mm. diameter

experienced in eliminating the personal equation so as to obtain satisfactory values for these two plates. The difficulty lies in the fact that the bulk of the galena went off at very low angles, while a few grains hung on till the angle was extremely large.

Grains that are finer than those shown upon Plate I.—namely, slime-sort which rises in currents from 0. to 1.26 millimeters per second—give a very wide space between the quartz and galena for 1 pound of water; but the angles converge and meet at about 8 to 10 pounds of water; and with these and larger quantities of water all the grains of both minerals go off the table together at a little less than 2° slope.

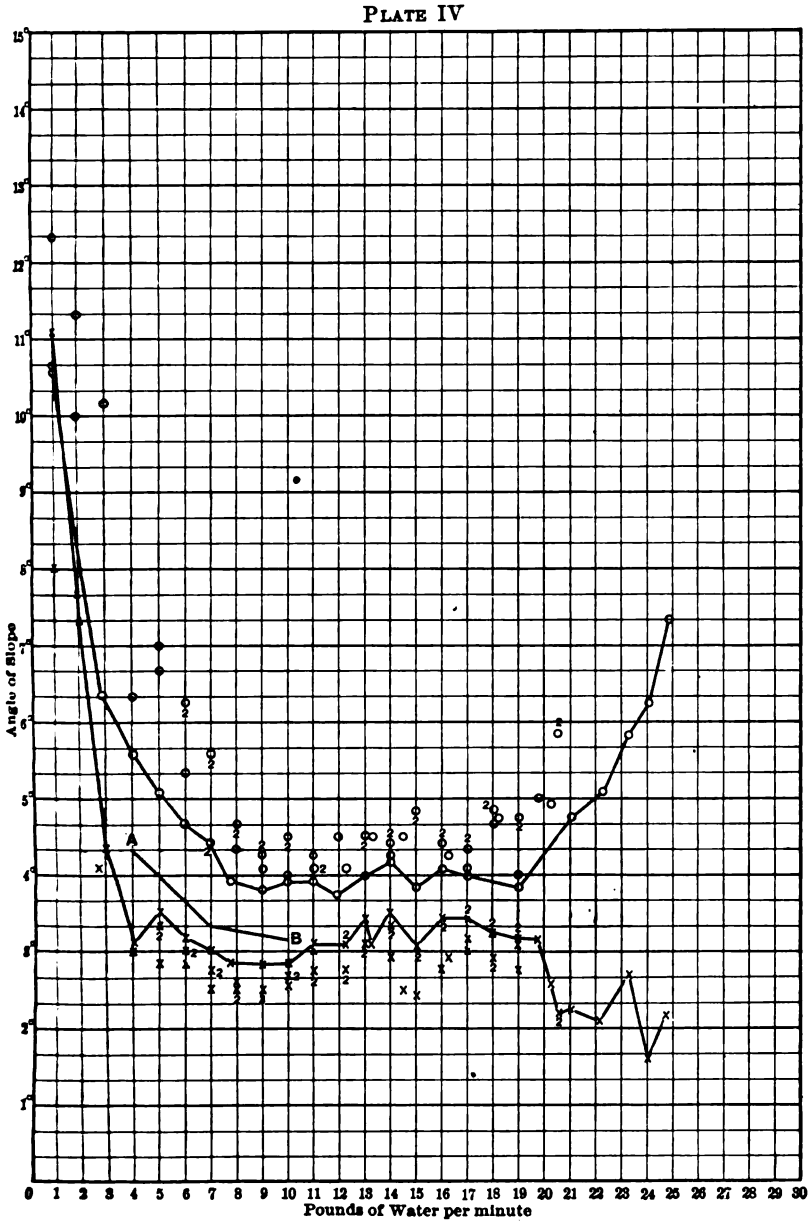
Tables III. and IV. are taken from the curves and show the minimum galena finish-angles and the maximum quartz finish-angles of the three experiments in each case.

TABLE III.—*Galena Finish-Angles at Which All the Grains Move.*

Pounds of Water per Minute on 2 Feet of Width.	Millimeters per Second of Current Which Lifts the Particles. 1.26	2.51	5.05	14.68	40.37	50.08	70.34
	Millimeters per Second of Current in Which the Particles Fall. 0	1.26	2.51	10.01	30.12	40.37	60.09
	Galena Finish-Angles (Minimum of Three Trials in Most Cases).						
1	15°20′	.....	9°00′	12°45′	10°40′	.....	.....
2	12°20′	.....	11°00′	9°30′	10°00′	.....	.....
3	13°20′	7°50′	8°15′	7°20′	6°15′	8°35′	9°55′
4	7°50′	10°50′	6°15′	7°10′	5°30′	7°30′	9°00′
5	.....	6°25′	5°50′	5°50′	5°05′	7°10′	7°00′
6	3°05′	6°00′	4°15′	5°15′	4°40′	5°55′	7°30′
7	.....	7°35′	3°55′	5°35′	4°25′	5°25′	5°35′
8	.....	4°30′	5°00′	4°45′	3°55′	4°55′	6°10′
9	.....	4°40′	5°30′	4°35′	3°55′	4°45′	5°40′
10	0°50′	.....	.....	4°15′	4°00′	4°20′	5°05′
11	.....	.....	.....	4°00′	4°00′	4°05′	5°10′
12	.....	.....	.....	4°00′	3°50′	4°05′	4°15′
13	.....	.....	.....	4°05′	4°00′	4°15′	4°40′
14	1°05′	.....	.....	4°15′	4°10′	4°20′	4°50′
15	.....	.....	.....	4°15′	3°50′	4°10′	4°40′
16	.....	.....	.....	4°20′	4°05′	4°10′	4°25′
17	.....	.....	.....	4°05′	4°00′	4°25′	4°15′
18	0°50′	.....	.....	4°30′	4°40′	4°35′	4°15′
19	.....	.....	.....	5°00′	3°50′	4°40′	4°15′
20	.....	.....	.....	4°30′	4°55′	5°05′	4°15′
21	.....	.....	.....	4°45′	4°45′	4°50′	4°45′
22	.....	.....	.....	5°05′	5°05′	5°00′	4°50′
23	.....	.....	.....	4°40′	5°50′	5°20′	5°05′
24	.....	.....	.....	5°30′	6°10′	5°30′	5°10′
25	.....	.....	.....	5°35′	7°20′	5°30′	4°55′

Water-Films.—Plates VII., VIII. and IX. are examples selected out of the fifteen plates which were drawn to get average

PLATE IV



Grains which fall in an upward current of 30.12 mm. per second and rise " " " " " 40.37 " " "

O – Galena of .1306 mm. diameter x – Quarts of .3416 mm. diameter

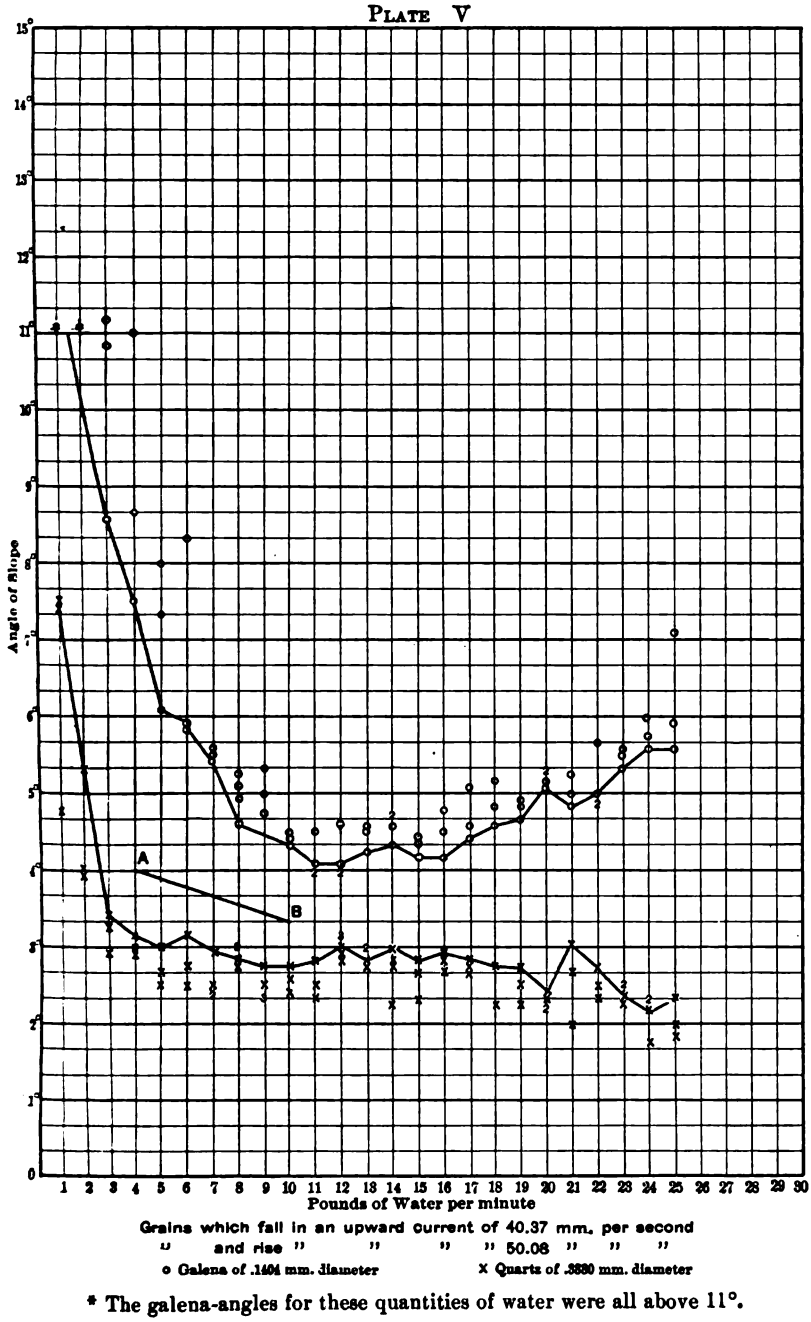
values for the thickness of water-film. Each plate represents a single rate of water-feed, and has all the film-thicknesses recorded for that weight of water. From these observations average curves are drawn, and each point on the curves represents: (1) the thickness of film (abscissa) which the given quantity of water will have with any angle of slope (ordinate); (2) where waves form, the angle at which the tendency begins is shown; and (3) the average heights of both crest and trough are given. These film-thicknesses are considered important aids to interpreting the effects of the finish-angles of Tables III. and IV. and Plates I. to VI. The complete set of values of these film-thicknesses is given in Table VI. Curves of water-thickness are also given on Plate X. These latter values are calculated from the figures given in Table VI. by assuming

TABLE IV.—*Quartz Finish-Angles at Which All the Grains Move.*

Pounds of Water per Minute on 2 Feet of Width.	Millimeters per Second of Current Which Lifts the Particles. 1.26	2.51	5.05	14.68	40.37	50.08	70.34
	Millimeters per Second of Current in Which the Particles Fall. 0	1.26	2.51	10.01	30.12	40.37	60.09
	Quartz Finish-Angles (Maximum of Three Trials in Most Cases).						
1	4°	3°20′	1°55′	3°45′	11°05′	7°30′	8°40′
2	3°40′	2°35′	2°25′	3°25′	7°30′	5°20′	5°25′
3	2°30′	2°50′	2°05′	3°15′	4°50′	3°25′	4°10′
4	2°40′	2°30′	2°20′	3°00′	3°50′	3°10′	3°15′
5	.....	2°20′	2°00′	2°35′	3°30′	3°00′	3°00′
6	1°05′	2°35′	2°00′	2°40′	3°10′	3°10′	2°45′
7	.....	2°05′	2°05′	2°35′	3°00′	2°55′	2°50′
8	.....	2°15′	2°10′	2°40′	2°50′	2°50′	2°35′
9	.....	2°15′	2°10′	2°45′	2°50′	3°05′	2°50′
10	0°50′	2°15′	2°00′	2°25′	2°50′	2°45′	2°55′
11	.....	2°00′	2°15′	2°45′	3°05′	2°50′	2°40′
12	.....	1°35′	2°20′	2°05′	3°05′	3°00′	2°40′
13	.....	2°00′	2°10′	3°15′	3°25′	2°50′	3°10′
14	1°05′	2°05′	2°00′	2°40′	3°30′	2°55′	3°15′
15	.....	1°55′	1°40′	2°05′	3°05′	2°50′	3°05′
16	.....	1°40′	1°35′	2°00′	3°30′	2°55′	3°10′
17	.....	2°45′	2°00′	1°50′	3°30′	2°50′	3°25′
18	0°50′	1°20′	1°40′	1°40′	3°15′	2°45′	3°25′
19	.....	1°35′	1°35′	2°05′	3°10′	2°45′	3°15′
20	.....	2°15′	1°45′	2°30′	3°10′	2°25′	2°45′
21	.....	1°50′	1°50′	2°10′	2°15′	3°05′	3°30′
22	.....	1°30′	1°20′	1°40′	2°05′	2°45′	4°00′
23	.....	1°15′	1°25′	2°05′	2°40′	2°20′	3°05′
24	.....	1°45′	1°50′	1°35′	1°35′	2°10′	2°45′
25	.....	1°30′	1°55′	1°50′	2°10′	2°20′	2°40′

that the mean thickness of the film is one-third the way from the thickness of the trough to the thickness of the crest.

PLATE V

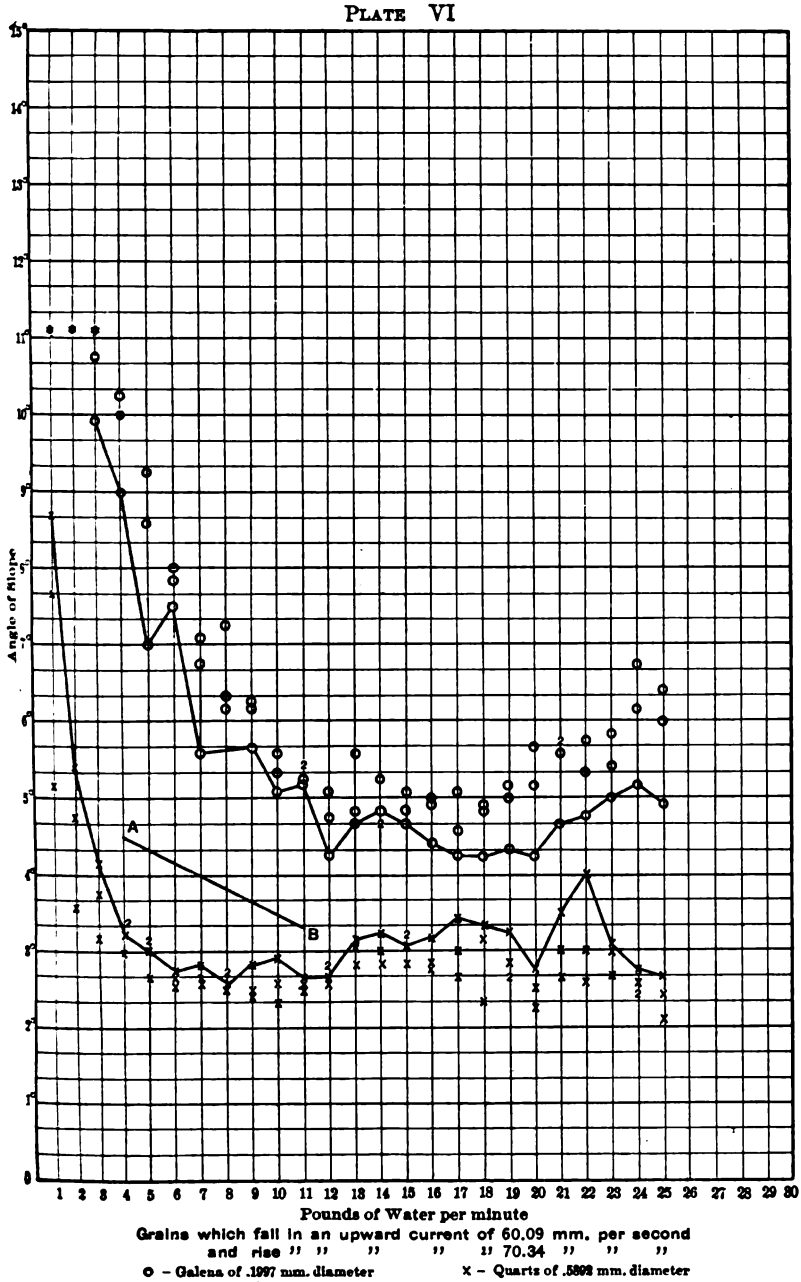


* The galena-angles for these quantities of water were all above 11°.

TABLE VI.—*Thickness of Water-Films in Inches.*

SLOPE OF TABLE.	POUNDS OF WATER PER MINUTE OVER A SURFACE 2 FEET WIDE.																											
	0.5		1		2		4		6		10		14		18		22		26		30		34		54	70	86	
	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.	Crest.	Trough.				
0°15′	.....	.100	.....	.085	.....	.070	.....	.065	.....	.067	.....	.130	.....	.150	.....	.150	.....	.170	.....	.195	.....	.165	.....	.200	.200	.219	.262	
1°	.....	.018	.....	.024	.....	.030	.....	.041	.....	.048	.....	.060	.....	.067	.072	.071	.....	.076	.....	.081	.....	.084	.....	.089	.100	.116	.138	
2°	.....	.014	.....	.019	.....	.024	.....	.032	.041	.037	.050	.047	.052	.049	.056	.053	.058	.056	.....	.059	.063	.062	.....	.062	.078	.089	.101	
3°	.....	.012	.....	.018	.....	.021	.030	.027	.039	.031	.047	.040	.048	.040	.050	.045	.052	.049	.055	.053	.055	.053	.....	.054	.067	.076	.084	
4°	.....	.011	.....	.017	.021	.020	.031	.024	.039	.029	.047	.035	.047	.035	.050	.042	.050	.045	.052	.048	.050	.047	.....	.050	.060	.066	.075	
5°	.....	.010	.....	.016	.021	.020	.031	.023	.039	.029	.047	.032	.047	.034	.050	.039	.049	.043	.049	.044	.046	.043	.....	.047	.054	.061	.069	
6°	.....	.010	.....	.015	.022	.019	.032	.023	.040	.028	.047	.029	.047	.033	.051	.036	.048	.040	.047	.040	.044	.041	.....	.045	.051	.058	.....	
7°	.....	.010	.....	.015	.022	.018	.032	.022	.040	.028	.048	.028	.....	.....	.....	.....	.047	.037	.045	.038	.043	.040	.....	.043	.049	.....	.....	
8°	.....	.010	.....	.014	.022	.017	.032	.022	.041	.028	.....	.....	.....	.....	.....	.....	.046	.034	.044	.036	.....	.....	.043	.042	.....	.....	.....	
9°	.....	.010	.....	.013	.024	.017	.032	.022	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
10°	.....	.010	.....	.012	.024	.016	.032	.021	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
11°	.....	.010	.013	.012	.025	.015	.032	.021	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
12°	.....	.009	.013	.011	.025	.014	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
13°	.....	.009	.013	.010	.025	.013	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
14°	.....	.009	.014	.010	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
15°	.....	.009	.015	.009	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
16°	.....	.009	.016	.009	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
17°	.....	.009	.016	.008	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
18°	.....	.....	.016	.008	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
19°	.....	.....	.017	.007	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
20°	.....	.....	.017	.007	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	

PLATE VI



* The galena-angles for these quantities of water were all above 11°.

TABLE VII.— *Velocity of Water-Current on the Table in Inches per Second.*

SLOPE-ANGLE.	POUNDS OF WATER FROM 2 FEET OF WIDTH.														
	0.5	1	2	4	6	10	14	18	22	26	30	34	54	70	86
0°15′	0.1	0.23	0.55	1.17	1.72	1.48	1.79	2.31	2.50	2.56	3.50	3.27	5.19	6.16	6.31
1°	0.53	0.80	1.28	1.85	2.40	3.20	4.02	4.88	5.58	6.17	6.87	7.34	10.39	11.62	12.45
2°	0.69	1.01	1.60	2.37	3.03	4.00	5.38	6.41	7.55	8.45	9.30	10.51	13.30	15.15	16.35
3°	0.80	1.07	1.83	2.71	3.38	4.56	6.26	7.36	8.50	9.41	11.09	12.05	15.45	17.75	19.70
4°	0.87	1.13	1.92	2.91	3.59	4.92	6.90	7.70	9.03	10.18	12.01	13.05	17.30	20.50	22.10
5°	0.96	1.20	1.92	3.03	3.59	5.19	7.08	8.05	9.41	11.10	13.10	13.90	19.22	22.10	23.90
6°	0.96	1.28	1.92	3.03	3.59	5.49	7.27	8.45	9.87	11.85	13.70	14.50	20.30	23.25	.....
7°	0.96	1.28	2.03	3.03	3.59	5.65	.....	.....	10.59	12.48	14.05	15.20	21.20	.....	.....
8°	0.96	1.37	2.03	3.03	3.59	.....	.....	.....	11.15	13.15	.....	15.55	.....	.....	.....
9°	0.96	1.48	2.03	3.03	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10°	0.96	1.60	2.14	3.17	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
11°	0.96	1.60	2.14	3.17	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
12°	1.07	1.60	2.27	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
13°	1.07	1.74	2.27	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
14°	1.07	1.74	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
15°	1.07	1.74	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
16°	1.07	1.74	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
17°	1.07	1.92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
18°	.....	1.92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
19°	.....	1.92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
20°	.....	1.92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

PLATE VII

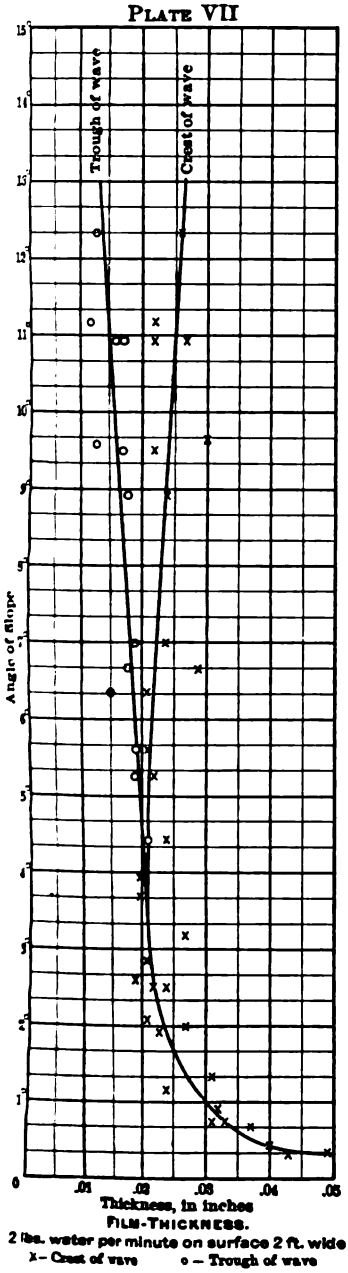
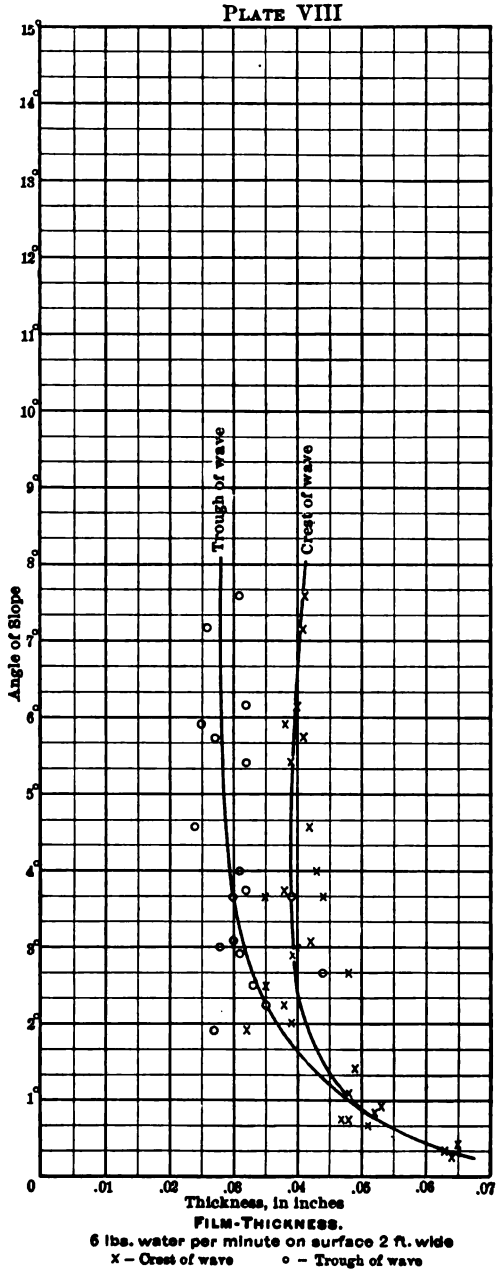


PLATE VIII



Velocity of the Current.—The velocity of the water-current (see Table VII. and Plate X.) is computed upon the assumption that the average thickness of film is one-third of the way between the thickness of the trough and that of the crest. These velocities will have a bearing upon the carrying-power of the water, and, therefore, upon the work of the slime-table.

Waves.—The tendency to form waves is best shown in Plate XI. The lines marked 10, 15, 20 per cent. and so on trace out that part of the field where the wave is elevated above the trough to a height of 10, 15, 20 per cent. and so on, of the thickness of the trough. Plate XI. was prepared from fifteen plates, of which Plates VII., VIII. and IX. are three, and from Table VI. This is an exceedingly interesting plate. It explains much that happens upon a table.

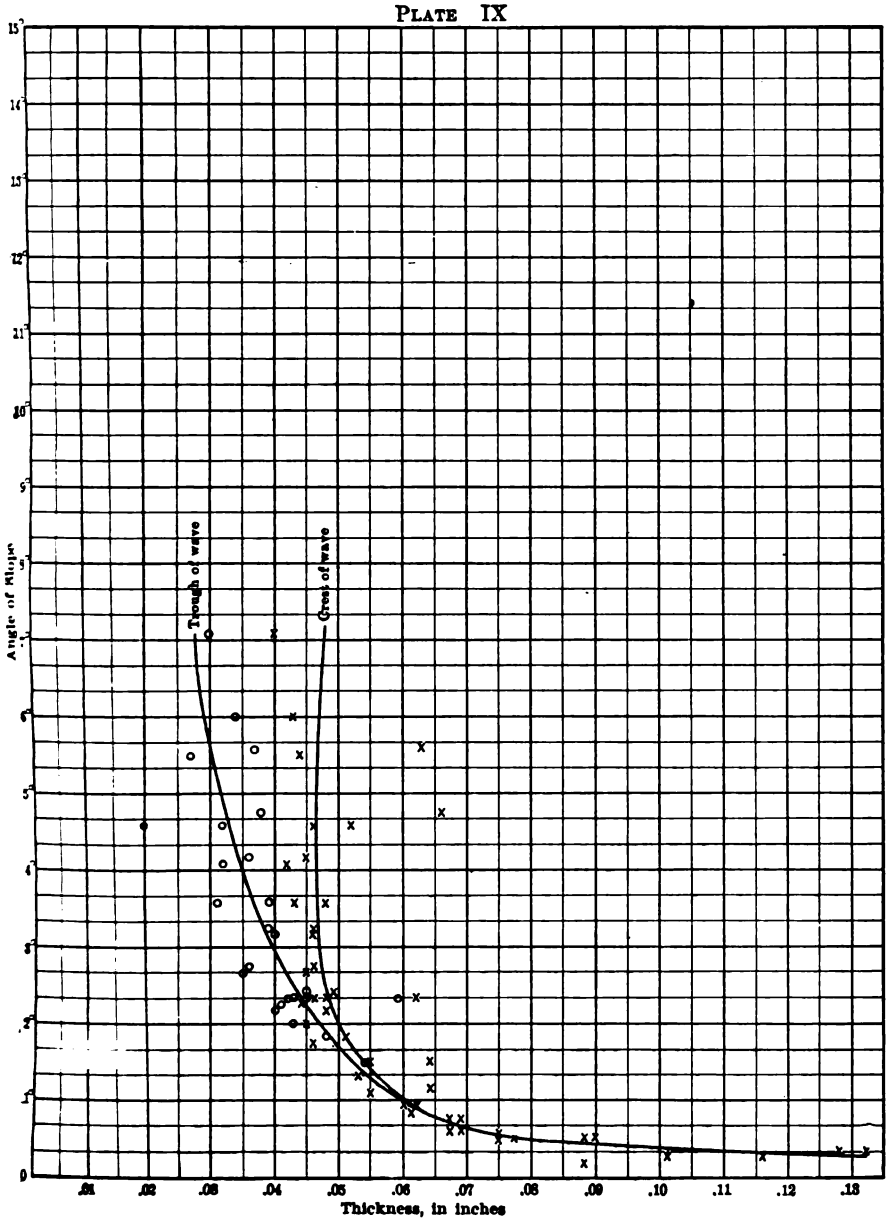
Conclusions.

Preliminary Classification.—Custom in this country has very nearly abolished classification of slimes. In some instances the overflow of the last hydraulic classifier is sent to a *Spitzkasten* apparatus with two or three pointed boxes, the product from each box going to its table; but it is more common to send these slimes to a distributing-tank, which sends whole slime to each of the tables, all of which are fed alike with a like quality of slime.

Is this wise? The author is of the opinion that it is not wise, for the following reasons: Practice employs 6 to 14 pounds, or an average of about 10 pounds of water per minute on every 2 feet of width. This quantity washes away with the tailings the whole of the first slime-sort shown in Tables III. and IV. on any slime-table that has a slope of 0.42 inch or more per foot, while, in fact, there is scarcely a table in the United States that has less than ¼-inch slope to 1 foot. Again, upon a glance across Table IV., it is noticeable that heavier slime-sorts are not suited to the same angle as lighter slime-sorts with the same water.

The Quartz- and Galena-Curves.—In Plates III., IV., V. and VI. the quartz-curve, passing from left to right, first drops somewhat, then bends round to nearly horizontal, diminishing a little as the water increases, and, except for the sudden fall at the left, it is nearly straight.

PLATE IX



THICKNESS OF FILMS

10 lbs. water per minute on surface 2 ft. wide.

x – Crest of wave o – Trough of wave

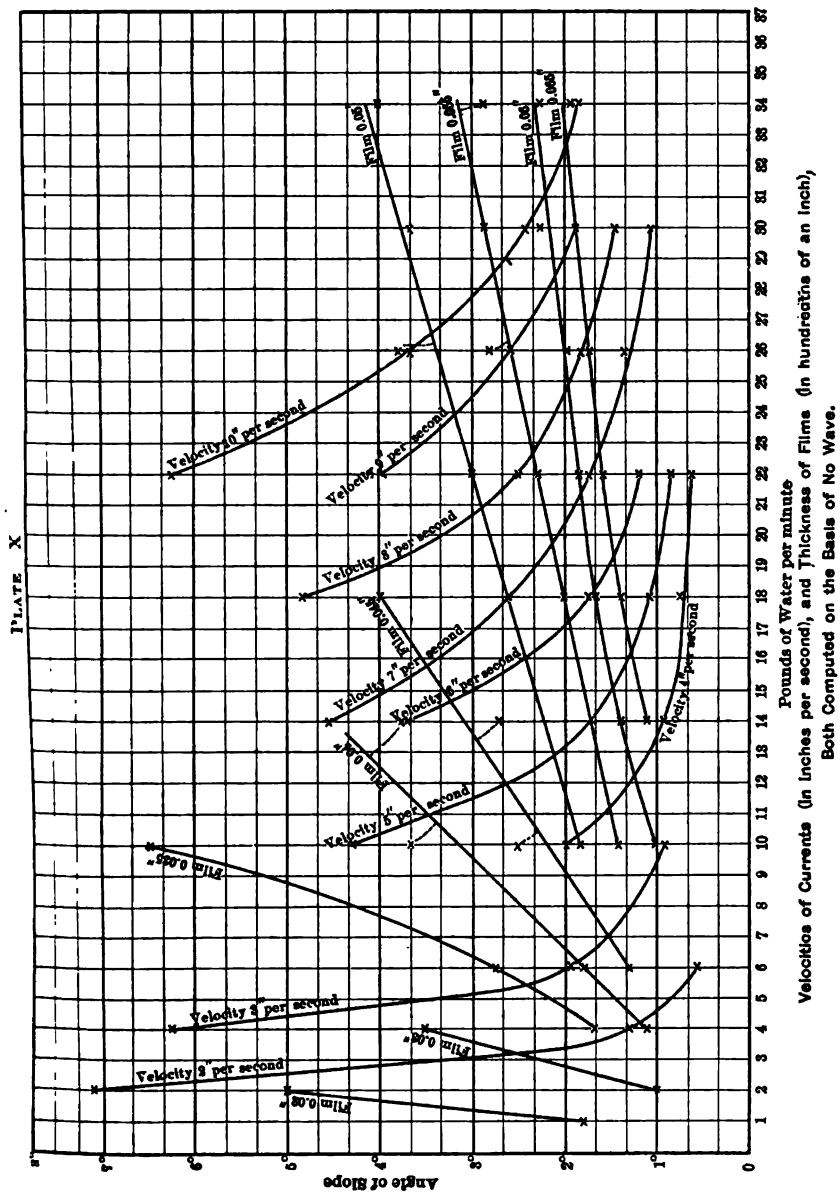
The galena-curve behaves very differently. It begins quite high up on the left, falls down rapidly as the water increases, then assumes a horizontal direction, and finally rises again to a considerably increased angle with large quantities of water.

Two questions arise, both of which are of interest in ore-dressing: 1. What is the cause of difference between the quartz- and galena-curves? and 2. To what conclusions does it point?

First, as to the cause. Referring to Plate XI., one sees that the tendency to form waves almost exactly corresponds to the galena curves (see Plates III., IV., V., VI.), that is to say, where the waves form freely, there the galena moves with a low angle. The higher the wave, the lower the angle at which the galena moves. This tendency of the waves to hurry the galena off the edge of the table, under the conditions here considered, suggests harm to the table-work. The waves might not be a hindrance to the ore-dresser if the quartz were affected in the same way as the galena. This, however, is not the case; the quartz is all washed off the table at an angle of slope at which the waves are slight, or do not exist at all. On this account the quartz does not show a corresponding dropping of the angle with that shown by galena. It is quite fair to presume that, if some way could be devised of killing the waves, galena would have a curve far up in the steep angles and parallel with the quartz all across.

Secondly, to what conclusions does this curious phenomenon point? The wider the interval between the quartz and the galena, the better would the field appear to be for treatment. If, for instance, all the quartz can be washed off while all the galena can stay at rest upon the table, then the results should be of the best. Now, there are two areas where the curves recede from each other. One is in the small water-quantities (2 pounds to 6 pounds), the other is out towards 20 and 25 pounds. These two areas point out two places of good treatment. The 25-pound area may be ruled out on account of the large quantity of water needed, unless there be special cases or special designs made which will enable the large quantity of water to treat also a large quantity of ore and so justify its adoption. We are left then for the most work to the small water area from 2 to 6 pounds of water, more or less, and to quartz-angles that are a little steeper in this part of the field.

Criticism may be directed towards the fact that these tests have been made with a plane table, whereas most of the wash-



ing is done upon convex conical tables. In reply it may be said that the convex table has a speed of flow and a thickness of film diminishing from the center towards the circumference,

but the amount of change is very slight for 1 foot of distance when near the circumference of the table. If, therefore, the water be adjusted to give the quantity desired for the space, say from 1 to 2 feet from the margin, the conditions should be found there to be practically the same as on the plane table. No matter what may have happened higher up the slope, when the grains reach this foot of surface they should obey the law and stop or roll according to their size, the degree of slope and the amount of water.

Slope-Angles and Water-Quantities of Best Treatment.—To bring the results down to a practical basis a line A B of "best treatment" has been drawn upon each of the Plates I. to VI. Somewhere upon this line, in the author's opinion, will be found the best treatment for the particular slime-sort. The water-quantities and slope-angles for the two points A and B are given in Table V.

TABLE V.

Particles fall in current of	Particles rise in current of	POINT A.		INTERMEDIATE POINT.		POINT B.	
		Water.	Slope.	Water.	Slope.	Water.	Slope.
Millimeters.	Millimeters.	Pounds.		Pounds.		Pounds.	
0.	1.26	1	8°	.....	.....	4	3°
1.26	2.51	2	6°	.....	.....	5	3°20′
2.51	5.05	2	5°	.....	.....	5	3°20′
10.01	14.68	2	5°	.....	.....	7	3°20′
30.12	40.37	4	4°20′	7	3°20′	10	3°00′
40.37	50.08	4	4°	.....	.....	10	3°20′
60.09	70.34	4	4°30′	.....	.....	11	3°20′

These slime-sorts can be grouped together into three classes:

I. Particles which fall in 30 mm. and rise in 70 mm. per second current.

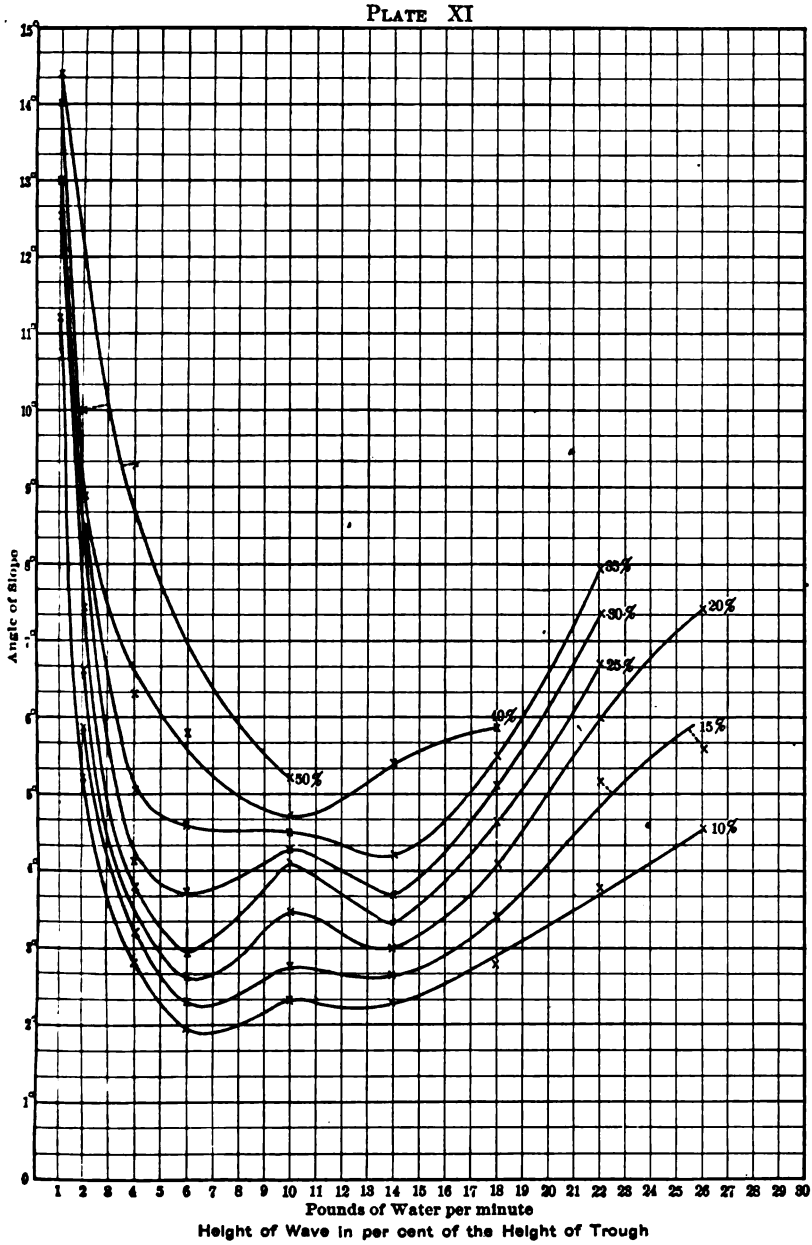
II. Particles which fall in 1.26 and rise in 30 mm.

III. Particles which rise in 1.26 mm.

If these three great groups are adopted they will probably each be found to contain particles sufficiently near each other in behavior to be treated alike.

The plates clearly suggest a small quantity of water and steeper slopes for finer products, and a large quantity of water and gentler slopes for coarser products. The author has already seen, in the course of these experiments, indications that, in

PLATE XI



Height of Wave in per cent of the Height of Trough

products which are well sorted, galena and quartz are susceptible of separation by large water-quantity and flat slope, even when the quartz is as large as 10-mesh.

Present and Proposed Practice Compared.—Present practice uses tables ranging in slope from $\frac{3}{4}$ inch to $1\frac{1}{2}$ inches to the foot. It also uses quantities of water for 2 feet of width that range from 6 pounds to 14 pounds. If we put the average at 10 pounds and consider a table with $1\frac{1}{2}$-inch slope to the foot, then comparing Tables III. and IV. and also Plates I. to VI. we see that both the quartz and the galena are moving for the edge of the table. They are, in fact, racing and the quartz is winning the race.

A table with $\frac{3}{4}$-inch slope to the foot, and with 6 to 7 pounds water would be, for some slime-sorts, within the author's figures; and the galena would be largely at rest upon the table, with the quartz departing from it. We may compare the two methods then by the two words "racing" and "departure." Most of the tables at present used are working in the portion of the field where the quartz and galena curves are closest together, and would seem therefore least advantageously placed. They are able to do this, because they are working upon the "racing" principle. The method of departure which the author suggests selects its area where the curves are far apart, where the galena may easily be put to rest upon the table and the quartz roll away and leave it behind.

Thus far, in this comparison, the ground has been taken that slimes should be sorted with a positive hydraulic and that an angle slightly larger than the quartz finish-angle is the right one to use, and the inference might be drawn that the writer condemns all the tables at present used. It would not be fair to the present practice or to the writer to close this paper without further reference to this point. The present plan uses the *Spitzkasten*, or the settling-tank, to distribute pulp to the tables. With the former the coarser slime-sorts have with them finer slime-sorts which the tables throw wholly away; with the latter all the tables have fine slime-sorts which they throw away. Both systems usually send to waste the overflows, which may contain quantities of valuable ore.

The mill-man has had in his possession no remedy for this throwing away of so much fine ore. In the first place, he has

had no device for preventing the great volume of water from coming to this point, and no better way of handling it has been devised than to settle out what could be settled quickly, to treat it on a table, saving what could be got and to let the rest go.

A proposition, then, which asks for positive hydraulic to really classify the slimes, and does not diminish the quantity of the water until the last slime-sort is reached can hardly find favor unless it also suggests a way for diminishing the great quantity of water coming to this point. The author thinks this can be done by diminishing the number of ordinary hydraulic classifiers. They are all great diluters, and simply add water at that point which is to rob the treatment of large quantities of fine slimes, which must be lost in consequence.

Why cannot the European limit of $1\frac{1}{2}$ mm., or even finer holes, for the last trommel be adopted? One mill now operating in the United States is sifting down to $\frac{1}{2}$ mm. and jigging the last-sized product with very clean tails—cleaner, in the author's opinion, than those of jigs which are treating separator-spigots, except, perhaps, those run on the Lake Superior basis, with finishing jigs below.

The author believes that if our mills should use more trommels, carrying the sifting to a much finer point than at present, the tailings of the fine jigs would be cleaner and the systematic concentration of slimes possible.

The overflow of the *Spitzkasten*, or the distributing-tank, should not go to waste. It should all go over tables with suitable slope, water-quantity and speed.

Table-Speed.—It will be clear that with the gentler slopes and water-quantities proposed by the author the tables must also move more slowly to give time for the quartz to roll off. Instead of the present speed of one revolution per minute for a 17-foot table the speed must be diminished to one revolution in three minutes, five minutes or even more, as the work in hand demands.

The more the subject is studied the more intricate it seems to be; and when one hears that the slime-table has been tried at this place and at that, has proved a failure and has been thrown out, one is inclined to think that its principles were not understood, and that with a better knowledge of the apparatus such cases would be rare.

I wish to give credit to Messrs. William A. Tucker, Charles D. Demond and B. Stoughton for the good work they have done in the experimental part of the preparation of this paper.

The Distribution of the Precious Metals and Impurities in Copper, and Suggestions for a Rational Mode of Sampling.

BY EDWARD KELLER, BALTIMORE COPPER WORKS, BALTIMORE, MD.

(Chicago Meeting, February, 1897.)

In order to conduct intelligently the sampling of copper of various forms and grades, solid or in molten furnace-charges, a knowledge of this subject is essential. Yet figures and tests have been persistently withheld from publication, and the books at our command give us little or no information in this regard. This gap in our copper-literature has led to a series of experiments, the results of which are given below.

Nearly all gold- and silver-bearing copper, destined for shipment, is cast into the form of bars or pigs. For the sampler, the question arises, Does copper, so cast, remain homogeneous? If it does, sampling should present no difficulties; but, from experience, we know that this is not the case, and we are thus led to study the behavior of the precious metals and impurities in copper when the latter changes from the molten to the solid state.

For the purpose of the experiments here described, all available grades of copper were selected, and all of them were subjected to the same conditions, an important requisite for proper comparison.

The molten copper obtained directly from reverberatory- and blast-furnaces was cast in a cast-iron mould, the sides and bottom of which were 1 inch thick, and would, therefore, give the copper a fair length of time to chill. The inside dimensions of the mould, and, consequently, of the block of copper obtained, were: length, 10 inches; width, 9 inches; and depth, 5 inches.

Fig. 1 illustrates the method employed to obtain samples from these blocks. A was cut away, and the face so produced

on B was marked off over one-half of its surface into 1-inch squares (it being assumed tha the other half would give practically the same results); and by drilling through the center of each square, 3 inches deep, the individual samples, as indicated by S_1, S_2, S_3, S_4, S_5, etc., were obtained.

In this way each block should have yielded twenty-five samples. But, in the case of the unrefined copper, the uppermost inch of the block consisted mostly of blisters and some slag; while, in the case of the refined material, the surface, on set-

Fig. 1.

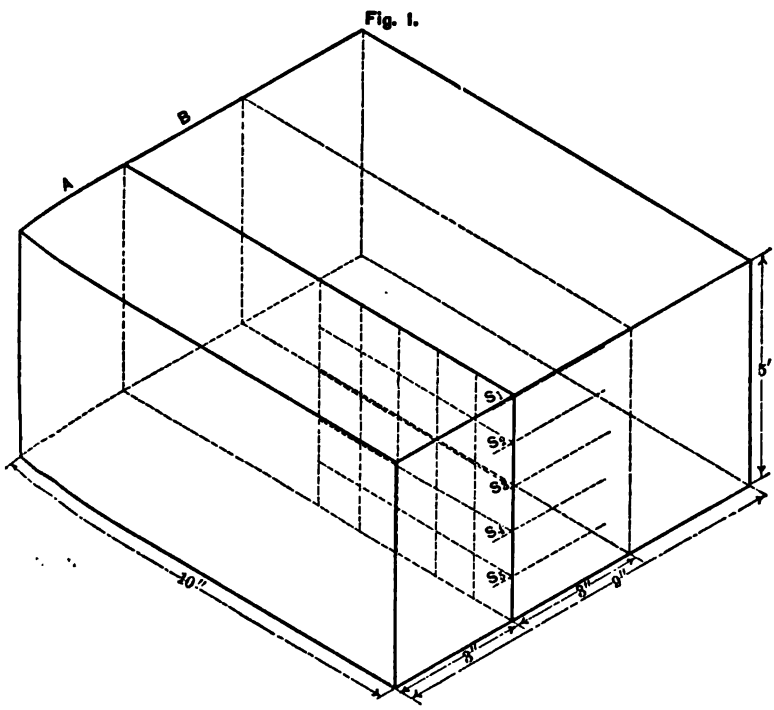


Diagram Showing Method of Taking Drill-Samples.

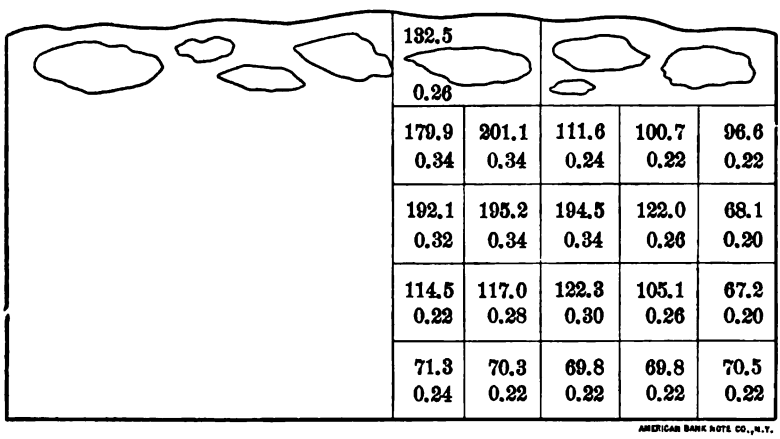
ling, had sunk nearly 1 inch. From every one of the blocks, therefore (with a single exception), only twenty samples were considered.

Fig. 2 shows the silver and gold determinations of a blister-copper for the twenty regular samples, as also of the uppermost crust, over the blisters. In this, as in the similar figures which follow, the amount of the precious metals is expressed in ounces per ton of 2000 pounds, the number for silver being in each couple the upper, and that for gold the lower.

For the purpose of general analysis of this block, samples were taken from the bottom, center and top of B (Fig. 1). They gave the following results:

	Bottom. Per cent.	Center. Per cent.	Top. Per cent.
Ag,	0.248	0.641	0.454
Au,	0.00068	0.00109	0.00089
Pb,	0.002	0.058	0.019
Bi,	0.005	0.055	0.024
Sb,	0.048	0.157	0.099
As,	0.034	0.108	0.074
Te,	0.004	0.027	0.019
S,	0.040	0.112	0.047
Total,	0.382	1.159	0.737

Fig. 2.



Silver and Gold Determined in a Block of Blister-Copper.

A glance at these figures convinces us that, in this case, a concentration or segregation of precious metals and impurities has taken place in a high degree towards—not the geometrical center, but—the center of solidification. This naturally lies above the geometrical center, more heat being given off at the bottom to the metallic mould than to the air on top—for which reason solidification proceeds more rapidly from the bottom.

Of the degree of concentration we can best form an idea by comparing each element of the bottom-sample with the corresponding element of the center-sample.

Dividing the value for the latter by the value for the former,

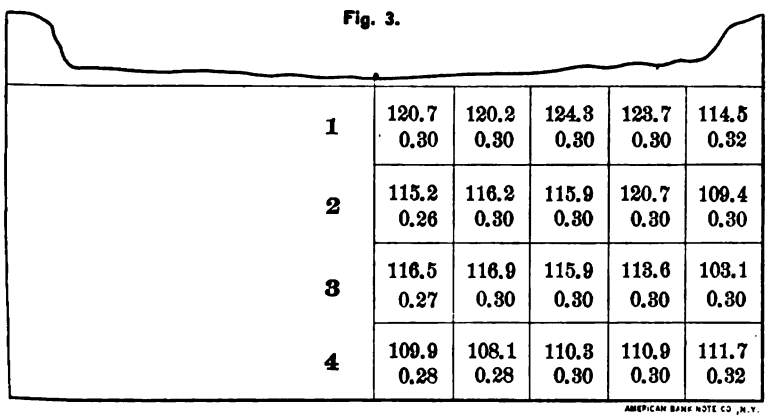
gives us the degree of concentration from bottom to center. We have accordingly:

Element.	Concentration.	Element.	Concentration.
Ag, ...	2.59	Sb, ...	3.27
Au, ...	1.60	As, ...	3.18
Pb, ...	29.00	Te, ...	6.75
Bi, ...	11.00	S, ...	2.80

The difference of behavior of the various elements is surprising.

That a sample which is not a correct average from a bar or pig of such copper might lead to great errors regarding its

Fig. 3.



Silver and Gold Determined in a Block of Refined Converter-Copper.

silver- and gold-contents, needs no further illustration. But copper-determinations also would be far from correct if made on a sample from any one part of such bar or pig. This will become evident at once if we take the copper by difference (100 — amount of impurities) in the three analyzed samples:

	Bottom. Per cent.	Center. Per cent.	Top. Per cent.
Cu by difference, . . .	99.618	98.841	99.263

Fig. 3 represents a block of refined converter-copper, with silver and gold as found in the respective samples.

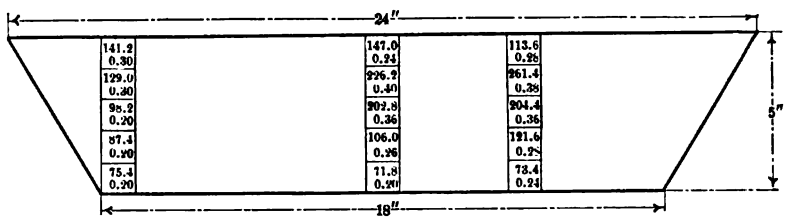
General analyses were made of Sample 1, Samples 2 and 3 combined, and Sample 4, with the following results:

	1	2 and 3	4
	Oz. per ton.		Oz. per ton.
Ag,	120.7		109.9
Au,	.30		.28
	Per cent.	Per cent.	Per cent.
Bi,	0.0026	.0017	.0013
Sb,	0.057	.060	.051
As,	0.022	.021	.014
Te,	0.008		.0056
O,	0.293		.200
(Cu_2O),	2.687		1.800

Among copper-metallurgists it is customary to give the amount of cuprous oxide instead of the oxygen. Both figures are, therefore, here inserted.

It will be noticed that the concentration of the precious metals and impurities in this second example is analogous to the first. They have moved from the sides and bottom towards

Fig. 4.



Silver and Gold Determined in an original Converter-Pig; Same Kind of Copper as Fig. 3.

the center and top, but the degree of concentration is very much reduced. This may easily be explained by the fact that the refined copper contains a much smaller amount of impurities than the blister-copper, and consequently a higher melting-point and a higher conductivity for heat. Both these properties favor a more rapid solidification, and thus shorten the time during which the molecules can move towards the center.

Fig. 4 represents an original converter-pig, of the kind of copper from which the previous sample was derived. The pig was drilled vertically through the center. The values found for silver and gold show no variation, as to distribution, from the two other samples already illustrated.

Fig. 5 represents a refined copper, in which the amounts of silver and gold, as well as the impurities, are considerably

higher than in those represented by Figs. 2 and 3. In this instance, Samples 1, 2, 3 and 4, were analyzed as follows:

	1	2	3	4
	Oz. per ton.	Oz. per ton.	Oz. per ton.	Oz. per ton.
Ag, . . .	270.4	309.4	350.0	353.0
Au, . . .	2.80	3.08	3.36	3.32
	Per cent.	Per cent.	Per cent.	Per cent.
Pb, . . .	0.204	0.248	0.351	0.392
Sb, . . .	0.163	0.185	0.211	0.217
As, . . .	0.134	0.151	0.197	0.212
Te, . . .	0.0125	. . .	0.015	0.0195

Here again we see differences in the quantities of the elements in each sample; that is, concentration. But if we com-

Fig. 5.

1	270.4 2.80	322.0 3.08	328.0 3.16	341.4 3.24	362.8 3.32
2	309.4 3.08	338.4 3.24	360.2 3.32	357.0 3.28	368.6 3.36
3	350.0 3.36	351.8 3.32	358.0 3.36	353.4 3.32	364.6 3.28
4	353.0 3.32	363.4 3.40	366.2 3.40	365.2 3.40	360.0 3.36

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Silver and Gold Determined in a Block of Refined Copper.

pare these with the former samples we observe the astonishing fact, that now we have the maxima in the position previously occupied by the minima, and *vice versa*. In other words, we have a copper in which the conditions have been reversed. Instead of the precious metals and impurities concentrating towards the center, it is now the copper which takes that place, and leaves the former elements concentrated towards the outsides.

Fig. 6 represents a black copper from a blast-furnace, but showing all the characteristics of a blister-copper. It is, in its relations to our subject, an analogon to the preceding sample.

Five samples were subjected to general analysis as follows:

	1	2	3	4	5
	Oz. per ton.	Oz. per ton.	Oz. per ton.	Oz. per ton.	Oz. per ton.
Ag,	.379	337.8	350.2	364.4	425.6
Au,	1.04	1.00	0.88	1.00	1.04
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Pb,	2.487	2.445	2.366	2.531	2.851
Bi,	0.400	0.360	0.360	0.310	0.420
Sb,	3.438	3.073	2.820	2.789	3.929
As,	0.705	0.716	0.630	0.627	0.706
Te,	0.892	0.768	0.814	0.856	0.882
S.,	0.538	0.467	0.462	0.487	0.469
Fe,	4.220	5.050	4.430	5.460	3.500

It will be seen here that sulphur and iron have behaved differently from the rest. Sulphur shows irregularities, while iron

Fig. 6.

1	379.0 1.04	387.6 1.00	378.0 1.00	427.2 0.92	507.8 1.00
2	337.8 1.00	362.2 0.92	384.0 0.92	399.8 1.04	489.4 1.04
3	350.2 0.88	355.8 0.88	361.0 0.92	392.6 1.04	474.6 1.04
4	364.4 1.00	374.4 1.00	381.2 1.00	402.8 1.08	488.8 1.04
5	425.6 1.04	424.0 1.04	432.8 1.04	458.0 1.00	505.6 1.00

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Silver and Gold Determined in a Block of Black Copper from a Blast-Furnace.

has concentrated towards the center; it has gone with the copper.

It will be observed also that the minimum amount of each element is not found in the same sample for all. In sample No. 2 we have the minimum for silver and tellurium, in No. 3 for gold and lead, and in No. 4 for bismuth, antimony and arsenic.

The blocks of copper represented by Figs. 2 and 3 were comparatively low in their contents of precious metals and impurities, while those in Figs. 5 and 6 were comparatively high.

The sample represented by Fig. 7 is low in precious metals and high in impurities, yet we find that the silver has concentrated in accordance with the latter class. A general furnace-

sample of this copper showed the following composition: Ag, 62.2, and Au, 0.14 ounce per ton; Pb, 0.78; Bi, 0.0035; Sb, 0.238; As, 0.052; Te, 0.0095; S, 0.796; and Fe, 0.17 per cent.

Fig. 8 is a photographic reproduction* of the face, B (Fig. 1), of the block of black copper represented in Fig. 6.

With the distinctness of a drawing it shows, by curved lines or cavities, each successive zone of solidification, and, by straight lines or cavities, the direction of solidification and concentration. The latter are perpendicular to the tangents of the curves, and converge towards the center of solidification, which is also clearly indicated much above the geometrical center, for reasons

Fig. 7.

Slag and Matte					
	59.6	59.2	60.3	60.4	61.3
	60.7	60.3	60.8	60.9	61.8
	60.5	60.8	61.2	61.3	61.6
	61.7	61.3	61.9	61.7	61.2

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Silver Determined in a Block of Black Copper of Lower Grade.

already given. The curved linear cavities were not readily discernible on the fresh cut, but were brought out plainly by rubbing with damp charcoal-powder.

The results thus obtained clearly prove the existence of two classes of copper-alloys. As already pointed out, in one class the foreign elements concentrate towards the center of solidification; in the other the copper concentrates to the center, while the foreign elements appear to have concentrated towards the outside.

Not a single element tested for has remained homogeneously distributed, but the degree of uneven distribution varies with each one.

From data given in Hofman's *Metallurgy of Lead*, pp. 244

* For the photograph the writer is indebted to Mr. William Keyser, Jr., of Baltimore.

and 245, I conclude that that metal shows the same characteristics in this respect as copper. This is confirmed by the article of A. Raht in the *Mineral Industry*, 1894, p. 414. Mr. Raht gives a few instances in which he shows that silver and gold in base bullion bars are highest at the bottom and sides, and lowest towards the top. He quotes Rosenlecher (from the *Berg-und Hüttenm. Ztg.*, October 5, 1894) as having found exactly the reverse, and accused the latter of strained figures, pet theories, etc. In my judgment there is no substantial reason for this inference. Undoubtedly Rosenlecher is as correct as Raht, and Raht as correct as Rosenlecher. The experiments of either covered but a narrow field, and neither discovered what the other did.

Very likely many alloys behave similarly, and homogeneity cannot be found in any of them.

Edward Matthey, in *Nature*, No. 1394, vol. liv., page 258, calls attention to the fact that bars of gold alloyed with zinc and lead show very uneven distribution, gold having concentrated to the center and bottom, gravity being one cause.

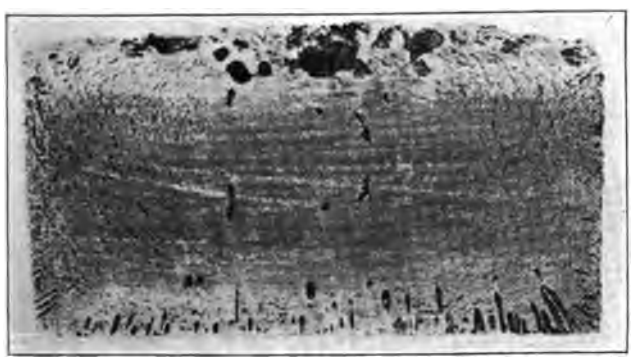
To overcome difficulties in sampling such material, Mr. Matthey looks for a solvent, and finds silver to be the medium. His figures cannot be pronounced entirely conclusive, and the method he proposes for obtaining reliable samples is very circuitous, to say the least, although the evil to be overcome is thereby minimized.

With reference to the practical question of sampling, we must draw from the indubitable facts stated the conclusion that it is almost impossible to obtain, from bars and pigs of the usual unsuitable dimensions, by drilling, punching or chipping samples of satisfactory accuracy. As we have seen, concentration takes place from or towards every surface. Hence, in such bodies of metal, what might appropriately be called a core is formed in the center, which is enriched or impoverished, as the case may be, and which does not conform in shape to the outer surface. It is for this reason so difficult to obtain aliquot parts of the different portions of a bar or pig. The root, then, of the difficulties of sampling, so often complained of, lies in the irrational shape into which the various metals are cast.

Another conclusion to be drawn from observed facts is the following:

When we take from a molten mass of copper a sample by

FIG. 8.



Face B (Fig. 1) of the Block of Black Copper shown in Fig. 6. The border-line outside of the photograph is not part of the face.

means of a ladle, the latter must be so hot as not to allow the forming of any solidified metal (sculling), for should this happen, the remaining liquid portion, which is to constitute the sample, would be for one class of copper too high in its contents of precious metals, and for the other class too low. If, however, we pour from a ladle, in which the whole metal is liquid, a so-called shot-sample, there is no assignable reason why it should not be a correct sample. When large samples of shot are taken from charges, which are supposed not to be homogeneous, they must, of course, be remelted, and a smaller sample taken in the same way, or a sample-plate must be cast, for which the rule described below must be observed; otherwise, the final sample may become erroneous.

In most smelting- and refining-works sample-bars or cakes are desired, and to produce these in the proper shape should be the duty of the sampler.

To eliminate all the difficulties inherent to pigs and bars, as above described, we only need to reduce one dimension; or, in other words, to cast a plate, the thickness of which is small compared to the other two dimensions. Of course, concentration in this plate is governed by the same law as in any other body: it takes place from or toward the center. But the concentration in the horizontal direction, from the sides, can extend no farther than a distance equal to the thickness of the plate, for when the solidification has proceeded that distance horizontally the plate has entirely solidified vertically. We thus have all around this plate a zone not wider than its thickness, where concentration has taken place just as in a bar or block; that is, both horizontally and vertically, and consequently we have one enriched and one impoverished band around the four sides. In the part of the plate enclosed by this zone we have had concentration in the vertical direction only, and the plate here consists, therefore, of what we may term layers of metal of different concentration. If we drill or punch through all of these layers we obtain a correct sample of the whole plate.

This seems to be the simplest solution of the sampling of any metal or bullion. It is difficult to see any reasonable objection to the casting into plates of appropriate size of any such material destined for shipment, of which subsequent sampling is required, and for which the remelting and taking of samples

from furnace-charges is impracticable. Uncertainty in sampling would thus be eliminated, and if beveled edges were given to the plates the handling of them would not be more difficult than that of bars.

Fig. 9 represents a plate of refined converter-copper 15 inches

Fig. 9.

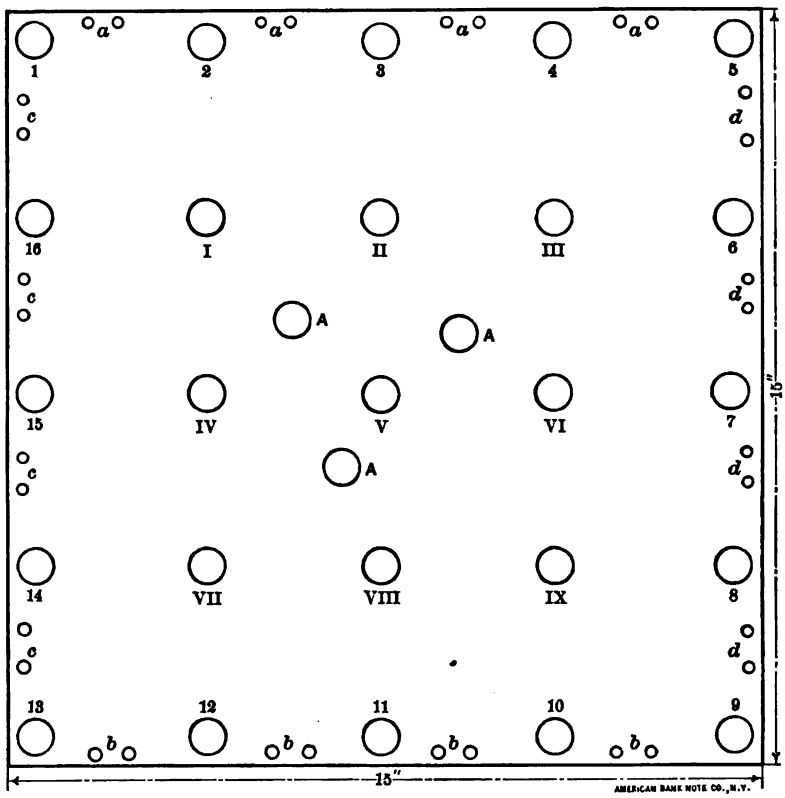


Diagram Showing Method of Taking Drill-Samples from a Plate of Refined Converter-Copper.

square by 1 inch in thickness. The holes designated by small letters were drilled with a $\frac{1}{4}$-inch drill, $\frac{1}{4}$ to $\frac{1}{8}$ inch from the edge of the plate. The drillings from each side were collected in one sample, making the four samples *a*, *b*, *c* and *d*.

The other holes were drilled with a $\frac{3}{4}$-inch drill. Those designated by Arabic figures are from $\frac{1}{8}$- to $\frac{1}{4}$-inch from the edge. Those with Roman figures, and the preceding ones, were drilled through the plate. The three marked A were

drilled ½-inch deep, and the drillings collected for the top-sample, then drilled through the plate for the bottom-sample.

The assays, in ounces per ton, were as follows:

Assays of Silver in Ounces Per Ton.

Sample.	Assay.	Sample.	Assay.	Sample.	Assay.	Sample.	Assay.
a,	105.	1,	99.2	I.,	98.4	A, top,	99.9
b,	100.7	2,	103.2	II.,	98.8	A, bottom,	98.0
c,	101.7	3,	104.2	III.,	98.9	Average,	98.9
d,	100.1	4,	99.8	IV.,	97.6		
Average,	101.9	5,	98.6	V.,	99.6		
		6,	97.8	VI.,	99.0		
		7,	98.5	VII.,	98.2		
		8,	99.3	VIII.,	99.4		
		9,	98.7	IX.,	99.5		
		10,	98.9	Average,	98.7		
		11,	99.4				
		12,	99.4				
		13,	98.9				
		14,	103.7				
		15,	98.5				
		16,	99.5				
		Average,	99.85				

In the discussion of the theory of a thin plate it was pointed out that there must exist around the edge a zone of higher and one of lower concentration than in any other part of the plate, due to the horizontal and vertical movement of the molecules in this zone. The assays have demonstrated the correctness of that statement, although the zone does not seem to be strictly defined. This undoubtedly is due to temperature-conditions of the mould and currents created in pouring the metal.

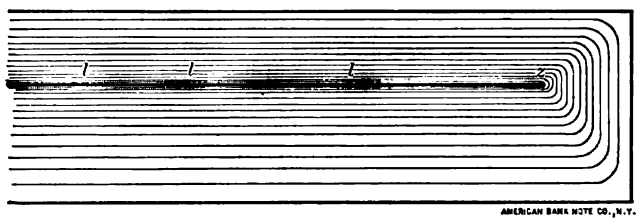
The vertical concentration in the plate, inside the edge-zone, is also clearly demonstrated by the top- and bottom-samples. These assays were repeated several times, while the rest were all made singly.

The samples and assays of the edge-zone are, of course, incorrect representatives of the true contents of the plate and the furnace-charge which it represents. The correct figure should be the average of assays I. to IX. inclusive, or 98.7 ounces, or the average of A, top, and A, bottom, 98.9. The two figures are very close together.

The plate from which the above figures are derived was cast

in a mould with very heavy bottom and thin sides. The solidification, therefore, took place much more rapidly from the bottom than from the sides and top. Fig. 10 illustrates this process, representing a vertical section through the edge and part of the plate. At *z* would be the highest or lowest point of concentration, according to the class of copper, the molecules meeting there from top, sides and bottom. At *l* would be the

Fig. 10.



Vertical Section Through Edge and Part of Plate (Fig. 9).

highest or lowest concentration of the layers, the molecules meeting from top and bottom.

For practical sampling plates from 8 to 10 inches square are

Fig. 11.

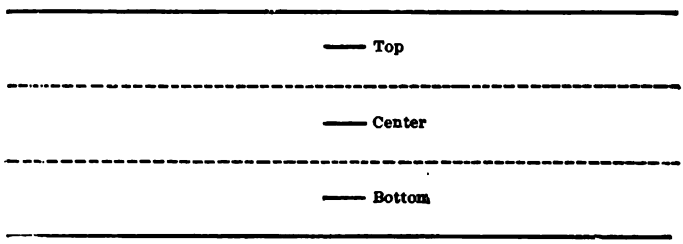


Diagram Showing Positions of Samples in a Liquid Charge of Copper.

of ample size, but they should not be more than 1 inch thick, and the drillings should be taken at least 1 inch away from the edge through the entire plate.

Having seen how a correct shot-sample, or a reliable sample-plate is obtained, we may now investigate how copper in its molten state is constituted.

The table below gives the results of tests, made for about three weeks, with furnace-charges of from 90,000 to 110,000 pounds of refined converter-copper. The copper having been poled, was, therefore, well mixed. Sample-plates were taken

in the middle of every third of the charge as it was tapped from the furnace, a practice adopted at these works. Fig. 11 illustrates the position of the samples in the charge.

Assays from Furnace-Charges.

	Oz. Ag.	Aver.	Oz. Ag.	Aver.	Oz. Ag.	Aver.	Oz. Ag.	Aver.
Top,	105.7		115.4		113.0		110.5	
Center,	106.5	105.8	113.9	115.0	111.8	111.6	109.7	110.0
Bottom,	104.3		115.7		110.1		109.8	
Top,	110.4		109.1		109.3		108.2	
Center,	110.2	109.9	108.9	108.7	108.4	108.8	108.1	108.1
Bottom,	109.1		108.1		108.7		108.1	
Top,	107.8		107.9		107.4		107.8	
Center,	107.9	107.9	108.6	107.7	107.9	107.7	108.2	108.0
Bottom,	107.9		106.6		107.9		108.0	
Top,	108.1		108.5		106.8		105.8	
Center,	108.1	108.1	108.8	108.5	106.9	107.0	106.8	106.4
Bottom,	108.2		108.1		107.4		106.5	
Top,	106.9		105.6		101.2		103.4	
Center,	106.0	106.2	106.2	105.9	101.4	101.3	101.8	102.2
Bottom,	105.7		105.8		101.4		101.4	

The greatest irregularities occurred during the first few days, and were due to the furnace being new, and the bottom seasoned with non-silver-bearing copper.

Each assay was made singly only, and some of the differences may be attributed, therefore, to possible experimental errors; although nearly all are within the usual allowable limits for this kind of work.

The following are the values in ounces per ton for silver and gold of three successive charges from a furnace of 30,000 pounds.

	Ag.	Average.	Au.	Average.
Top,	163.6		0.24	
Center,	163.2	163.1	0.24	0.24
Bottom,	162.6		0.24	
Top,	363.9		2.92	
Center,	363.3	363.3	2.96	2.95
Bottom,	362.5		2.96	
Top,	319.9		2.32	
Center,	320.1	320.0	2.36	2.35
Bottom,	320.1		2.36	

On the strength of these results, it is safe to say that any

charge of molten copper, once thoroughly agitated and mixed, is and remains uniform; and a sample taken from any part correctly represents the whole charge.

The subject treated above is not only of practical importance; it has also a very interesting scientific side.

It is, of course, nothing new to find uneven distribution of the elements in an alloy; but that this distribution should take place within very narrow limits of the quantities of these elements present, in two diametrically opposite ways, as shown by the preceding experiments, is a subject which has not been discussed, and which is decidedly perplexing to explain. No law known to the writer would apply to this remarkable phenomenon.

As shown by the analytical figures, the impurities, in the cases where they have concentrated to the center, have a far higher degree of concentration than in the case where they have gone towards the outside.

This is readily explained. In the first case the molecules move in the same direction as solidification takes place. They are, therefore, able to move and concentrate until the last particle of the metal is solid. By the increase of impurities towards the center, the melting-point of the alloy in that region is undoubtedly lowered, and thereby the solidification retarded, thus giving the elements more time to pursue their course.

In the case of concentration towards the outside, the elements move in the direction opposite to that in which solidification takes place, and by encountering the already solidified metal, their course is ended. The center being drained of impurities, its melting-point undoubtedly rises, accelerating the process of solidification, and reducing the time of motion and concentration.

That each element shows a different degree of concentration has already been pointed out. In explanation of this fact, it might simply be said that it is due to the differences of affinity; which, of course, is true. But for this term "affinity" between metals, we have as yet no expression in values or figures. We shall, therefore, attempt to find whether there is any relation between this and other physical properties of the atoms of the respective elements; and shall, for that purpose, first consider

the case of the copper represented by Fig. 2; the differences here being best defined.

In the following table the elements are arranged in the order of their degree of concentration. T is the melting-point in °C; A the atomic weight; D the specific gravity; and $V = \frac{A}{D}$ the atomic volume.

Concentration Compared with Physical Properties in the Copper of Fig. 2.

Element.	Concentration.	T.	A.	D.	V.
Pb	29.	332	206.4	11.38	18.1
Bi	11.	265	210.	9.82	21.4
Te	6.75	500	126.	6.25	20.2
Sb	3.27	437	122.	6.7	18.2
As	3.18	500	74.9	4.7*	15.9
S	2.80	115	31.98	2.04	15.7
Ag	2.59	967	107.66	10.5	10.2
Au	1.60	1037	197.	19.3	10.2
Cu	0.992	1057	63.3	8.8	7.2

Comparing the degree of concentration with the melting-point, we find a correspondence in a general way. The elements of low melting-point show a higher degree of concentration than those of high melting-point. Sulphur forms a very marked exception to this rule.

Between degree of concentration and either atomic weight or specific gravity, there seems to exist no direct relation. Gravity in this case, therefore, can be no factor governing the distribution of the elements.

Finally, in the last column, the values for the atomic volume unmistakably show a similar arrangement as those for concentration. From this fact a law can be formulated (in the way of suggestion) as follows: When on the solidification of a metal (copper) the small amounts of impurities segregate, or liquate, and consequently concentrate towards the center, the degree of concentration is greatest for those the atomic volume of which, compared with the atomic volume of that metal, shows

* This is the density given for amorphous arsenic. For the crystalline form it is 5.67, consequently V = 13.2.

the greatest difference. The influence of other properties (melting-point, etc.) would modify the law accordingly.

If the law expressed be true, we should then have, for that class of alloys, an alloy-series of the metals, in which the latter are arranged according to their atomic volume.

Metals in the Order of Atomic Volume.

Elem.	V.	Elem.	V.	Elem.	V.	Elem.	V.	Elem.	V.	Elem.	V.	Elem.	V.
Ni	6.7	Cu	7.2	Rh	8.6	Pt	9.1	Au	10.2	Mg	13.8	Tl	17.1
Co	6.9	Pd	8.2	Ir	8.6	Zn	9.1	Al	10.7	In	15.3	Pb	18.1
Fe	7.2	Ru	8.4	Os	8.8	Ag	10.2	Cd	12.9	Sn	16.1	Sb	18.2
												Bi	21.4

The theory thus deduced from an alloy of copper is in the main corroborated by an analysis of lead-bullion by Schertel (Hofman's *Metallurgy of Lead*, p. 244). Copper, iron and zinc, the atomic volumes of which differ most from that of lead, have concentrated in a higher degree than those which stand closer to the latter. Antimony, which is almost identical with lead in this property, has concentrated with that metal. Silver and bismuth do not appear to fall exactly in line.

In that class of coppers in which the impurities are found concentrated towards the outside, the above rule does not seem to apply, but owing to the small differences in the degree of concentration, and consequent large influence of possible experimental errors, it would be too venturesome to attempt the establishment of another rule. One fact observed in the copper represented by Fig. 6 seems also confirmatory of the theory

Concentrations Compared with Physical Properties in the Copper of Fig. 6.

Element.	Concentration.	T.	A.	D.	V.
Sb	1.39	437	122.	6.7	18.2
Pb	1.21	332	206.4	11.38	18.1
Ag	1.21	967	107.66	10.5	10.2
Au	1.18	1037	197.	19.3	10.2
Bi	1.17	265	210.	9.82	21.4
As	1.12	500	74.9	4.71	15.9
Te	1.08	500	126.	6.25	20.2
S	1.02	115	31.98	2.04	15.7
Cu	.987	1057	63.3	8.8	7.2
Fe	.79	2080	55.9	7.8	7.2

expressed, namely, the concentration of the iron with the copper, both metals having the same atomic volume.

The foregoing table gives the elements and their degree of concentration, as found in the above-mentioned copper, compared with the other physical properties.

The degree of concentration was obtained by dividing the values of the 5th sample by those of the 3d.

The material bearing on the subject here discussed is, as yet, too scant to permit the drawing of positive conclusions. It should always be remembered by investigators that to give their work full scientific value not only one or two elements in a combination of many, but the interrelations of all, must be observed and studied.

The Hand-Auger and Hand-Drill in Prospecting Work.

BY CHARLES CATLETT, STAUNTON, VA.

(Chicago Meeting, February, 1897.)

MUCH has been done of late years to facilitate preliminary stratigraphical investigations, and for examination at considerable depths and in a certain character of material the diamond-drill, in its general forms, certainly stands pre-eminent among the tools for such work. But there are large classes of deposits —indeed, I may say the larger proportion in which ordinary prospecting is done—which lie near the surface and are interbedded with soft and fine material. The method usually employed for prospecting these is by means of small shafts. These are sunk with considerable slowness and at considerable expense, depending largely upon the character of the material encountered, and are greatly interfered with by water. In this region the smallest are 3 by 4 feet, but they are usually considerably larger.

Having had occasion during the past year to do a large amount of prospecting work, I have found the use of a hand-auger and drill of immense value in this respect, and in my practice it has largely superseded, and always preceded, the other forms of investigation. I believe that the information desired was

thus acquired at one-fifth what it would have cost by any other method. I am therefore satisfied that a brief account of the work of this device will not be without interest to the members of the Institute.

Mr. Darton, of the U. S. Geological Survey, has referred to the excellent results obtained in his work by the use of a small auger, and a similar instrument has no doubt been used for prospecting purposes; yet it seems to be to no one's interest to manufacture them—at least I could learn of none on the market. I was therefore compelled to make them with such additions and modifications as experience showed to be necessary. The result may be summed up in the following description of a set of the tools.

1. An auger-bit of steel or Swede iron, with a steel point, twisted into a spiral, with an ultimate diameter of 2 inches and an ultimate thickness of blade of not less than ¼-inch. The point is found more effective when split. The length of the auger proper was gradually increased until about 13 inches was reached as the apparent maximum which could be used effectively. The 13-inch auger contains four turns. This was welded to the end of 18 inches of 1-inch wrought-iron pipe, on which screws were cut for connection.

2. One foot of 1⅜-inch octagonal steel, with a 2-inch cutting face, which is likewise welded on to 18 inches of pipe, cut for connections.

3. Ten feet of 1¼-inch iron rod, threaded at either end for connection with 1-inch pipe. When connected with one of the drill-bits this becomes a jumper for starting holes through hard material. It is also used when desired to give additional weight to the drill in going through rock below the surface.

4. Sections of 1-inch pipe and connections.

5. An iron handle, with a total length of 2 feet, arranged with a central eye for sliding up and down the pipe and with a set-screw for fastening it at any point.

6. A sand-pump, consisting of 1 or 2 feet of 1-inch pipe, with a simple leather valve and a cord for raising and lowering it.

7. Two pairs of pipe-tongs or two monkey-wrenches, with attachments for turning them into pipe-tongs.

8. Sundries: 25 feet of tape, oil-can, flat file, cheap spring-balance, water-bucket, etc.

The auger is used by two men, who, standing on opposite sides, turn it by means of the handle. The handle is also useful in giving a good purchase for starting the auger from the bottom of the hole, in opposition to the air-pressure, which is considerable. Enough water is continually used to just soften the material. Usually the auger brings up a small portion, which is dry and unaffected. Every few minutes, as the auger becomes full, it is lifted out, scraped off and replaced. The handle is moved up and tightened by means of the set-screw as the auger goes down. At every slight change of the material the depth and the character of the material are recorded.

When hard material is encountered the auger-bit is screwed off and the drill-bit screwed on, thus forming a churn-drill, which may be used for passing through the hard material, the auger being replaced when softer material is reached. The churn-drill is used by lifting it and letting it fall, turning it slightly each time. Its weight makes it cut quite rapidly. When the drill is used the muck is either worked stiff enough to admit of its being withdrawn with the auger, or it is extracted by means of the sand-pump or a hickory swab. In either case the material is washed and a sample is obtained of the stratum through which the drill is cutting. After washing all the material from one stratum the washed material is mixed and a sample is put into a bottle and labeled. In passing through "wash-ore" (*i.e.*, iron-ore consisting of small particles mixed in clay or other material that can be separated by washing) the material is brought up by the auger, put into a vessel with a perforated bottom (an empty tomato-can, having holes punched in the bottom and provided with a wire-handle, does very well), weighed by means of the spring-balance, washed and weighed again, and thus an approximate idea is obtained of the proportion of ore in the material, while a sample is secured for analysis.

As the mere recollection of the contents of the several holes cannot be safely depended upon, it is very important that an accurate record should be kept. To secure this is one of the difficulties encountered with ordinary foremen, though they are generally found to recognize accurately very slight changes of material. My foremen were provided with small vest-pocket note-books, and were soon trained to write down everything and to trust nothing to memory.

The holes were located as far as possible with reference to some natural object or to some other hole so located, and the distance therefrom was in all cases measured, not guessed at, the compass-direction being recorded. Every hole was marked with a substantial peg bearing its number and location.

In designating the holes upon the record care was taken to avoid all causes of confusion. Separate properties, or great natural divisions on large properties, were designated by letters, minor divisions by numbers. For instance, we would have as the first number at a certain location on a property "Location H, Drill-Hole 8000." The first number in the next natural division to the southward on the same property was designated "Location I, Drill-Hole 9000," and so on. By allowing 100 holes for the minor divisions of each locality, it was made possible to return and put in additional holes without confusion or conflict. With such a record it was possible to plat the location of the holes in plan and elevation, and give a visible representation of the "lay" of the strata intersected.

Of course the best work with such tools is done on soft material, but it is entirely practicable to go through hard material (a few feet of quartzite or flint, and many feet of ore being often encountered in a single hole), and the ability of this simple contrivance to go through interbedded layers of hard and soft substances makes it very efficient.

The cost per foot increases considerably with depths exceeding 50 feet, but at the greatest depth I attained (some 80 feet) I did not reach either its capacity or the limit of its economical use as compared with other methods.

Up to 25 feet, two men can operate it; from 25 feet to 35 feet, three men are necessary; from that to 50 feet, a rough frame, 15 feet to 20 feet high (costing something over $1.00), for the third man to stand on, is required. The frame can be moved from point to point. Above 50 feet it is generally necessary to take off one or two of the top-joints each time the auger or drill is lifted.

The following record of some holes and the time employed will be of interest. Labor was paid 90 cents per day of ten hours. A foreman at $1.25 could watch three or four holes. The only current repairs consisted in sharpening drills, and their amount, of course, depended upon the material:

Sample-Records.

"Location E., D. H. 1300. On top of divide south of Cold-Short Bank. Twenty-five feet east of old work."

	Feet.
Sand and gravel,	2
Yellow clay,	2
Yellow clay and little ore,	4
Solid ore,	5
White clay and ore,	3
	16

Two men 10 hours.

"D. H. 1381," etc.

	Feet.
Loose slide,	3
Blue clay,	7
Shale ore,	3
Wash ore (⅔ ore),	9
Shale ore,	3
Wash ore (⅔ ore),	15
	40

Two men 11 hours; 3 men 4 hours.

"D. H. 11,012," etc.

	Feet.
Light yellow clay,	12
Black flint,	½
Light yellow clay,	2½
White sand,	1
Solid sandstone,	2
	18

Two men 5 hours.

"D. H. 181," etc.

	Feet.
Soft sand,	3
Sandstone,	5
Yellow clay,	6
Soft red sandstone,	8
	22

Time not taken.

"D. H. 13,004," etc.

	Feet.
Sand and gravel,	1
Pink clay,	18
White clay,	10
	29

Two men 5 hours.

"D. H. 12,005," etc.

	Feet.
Light yellow clay,	14
Solid ore,	3
Yellow clay,	1
Solid ore,	5½
Yellow clay,	1
White clay,	1½
	26

Two men 6 hours.

"D. H. 802," etc.

	Feet.
Sand,	1
Shale ore,	4
Yellow clay and sand,	9
Sandstone,	5
	19

Two men 8½ hours.

"D. H. 8006," etc.

	Feet.
Red sand and sandstone,	19
Yellow clay with light streaks,	3
Light yellow clay,	8
Dark yellow clay,	6
Black clay,	2
Light yellow clay,	2
Light yellow clay with black streaks,	10
Light brown clay with white flint,	2
	52

Two men 15 hours; 3 men 4 hours.

"D. H. 8003," etc.

	Feet.
Sand and sandstone (drift),	12
Dark yellow clay and ore (¼ ore),	3
Light yellow clay with little ore,	7
Dark yellow clay and ore (⅓ ore),	9
Dark yellow clay with white flint,	3
Light yellow clay with small quantities of ore,	8
Dark brown clay with little ore,	8
Very black clay with small quantities of ore,	3
Dark yellow clay with ore (¼ ore),	2
Dark brown clay with small quantities of ore,	5
Yellow lifeless clay,	3
	63

Two men 5 hours; 3 men 25 hours.

The ore in most of the holes was brown hematite. In some of them it was manganese-ore. It varied very greatly in hardness.

The above records represent on the whole the most favorable conditions, the hard material being, as a rule, less than one-fourth of the entire depth, but it is interesting to note how the drill carries through the hard material and how the auger takes up the work on the other side.

In addition to the economy of operation, which means either a saving of money for the same amount of information, or the expenditure of the same amount of money and the acquisition of more information, there are other marked advantages in the use of such tools. The land is not injured for other purposes. Badly located holes (and there are always such) do not remain as conspicuous blots on the property. The holes can be located at points where it would be exceedingly inconvenient to put shafts. They can stand an amount of water which would add very greatly to the cost of shaft-work.

The following describes the way in which I have used the holes in a particular instance:

The iron-ore deposits of the Potsdam, as they are found in Virginia, are well known to many of the members of the Institute. It is sufficient to say here that the beds of ore, of more or less regularity, are interstratified with clays and decomposed shales, and are characterized by numerous folds, bends, secondary concretions and drift-material formed from the breaking down of the beds. The workings of the old iron-makers dot the horizon of these beds for several hundred miles.

In the examination of one of the old workings, consisting of a narrow open cut, which is about a thousand feet long, and must have been formerly some twenty feet deep, the natural supposition was that the old operators had commenced on the outcrop of the bed in place and had worked down as far as they could conveniently do so. But the important question was, Had they got through the beds, and was the remainder of the ore to be found in the direction of the dip, or was there still ore in the bottom of the cut? Holes put down at intervals along the bottom revealed no ore, while a line of holes, 50 feet apart, put down on the side of the cut in the direction of the dip, encountered, almost without exception, the beds of ore, and gave very positive information as to their average thickness, dip and composition.

From my experience of the past year with these drills I am

sure their more extended use would be of immense benefit to the mining industry. As one of our members has pointed out, a good thing that is not patentable is rarely brought to public attention, because it is to no one's interest to advertise it; but perhaps modifications might be invented increasing the convenience and efficiency of these already very effective tools, which would justify some one in making a specialty of them and bringing them to the attention of prospectors. Even without the protection of patents, I venture to believe, the manufacturer of an outfit of the character I have described, if properly advertised, would meet an existing and increasing demand and benefit the manufacturer as well as the user of the tools.

The Quality of the Boiler-Water Supply of a Portion of Northern Illinois.

BY JAMES A. CARNEY, AURORA, ILL.

(Chicago Meeting, February, 1897.)

THE quality of water for steam boilers is a serious question in Illinois, and has been the occasion of many investigations, undertaken either for the discovery of sources of a better supply, or for the purification of the supplies already at hand.

LOCATION OF SUPPLIES.

The sources of supply of which this paper treats lie along the right of way of the Chicago, Burlington and Quincy Railroad Company, and form a nearly straight line from Chicago, Illinois, on Lake Michigan, to Burlington, Iowa, on the Mississippi River. The supplies consist of surface-, ground- and deep well-waters; their locations are given in Fig. 1, which is adapted from a map of Illinois published by the Illinois Railroad Commissioners.

The influence of the geological formation on the water-supply of Illinois is fully treated in an article in the Seventeenth Annual Report of the U. S. Geological Survey by Mr. Frank Leverett, entitled, "The Water-Resources of Illinois." It is not the purpose of this paper to introduce a geological discussion; and

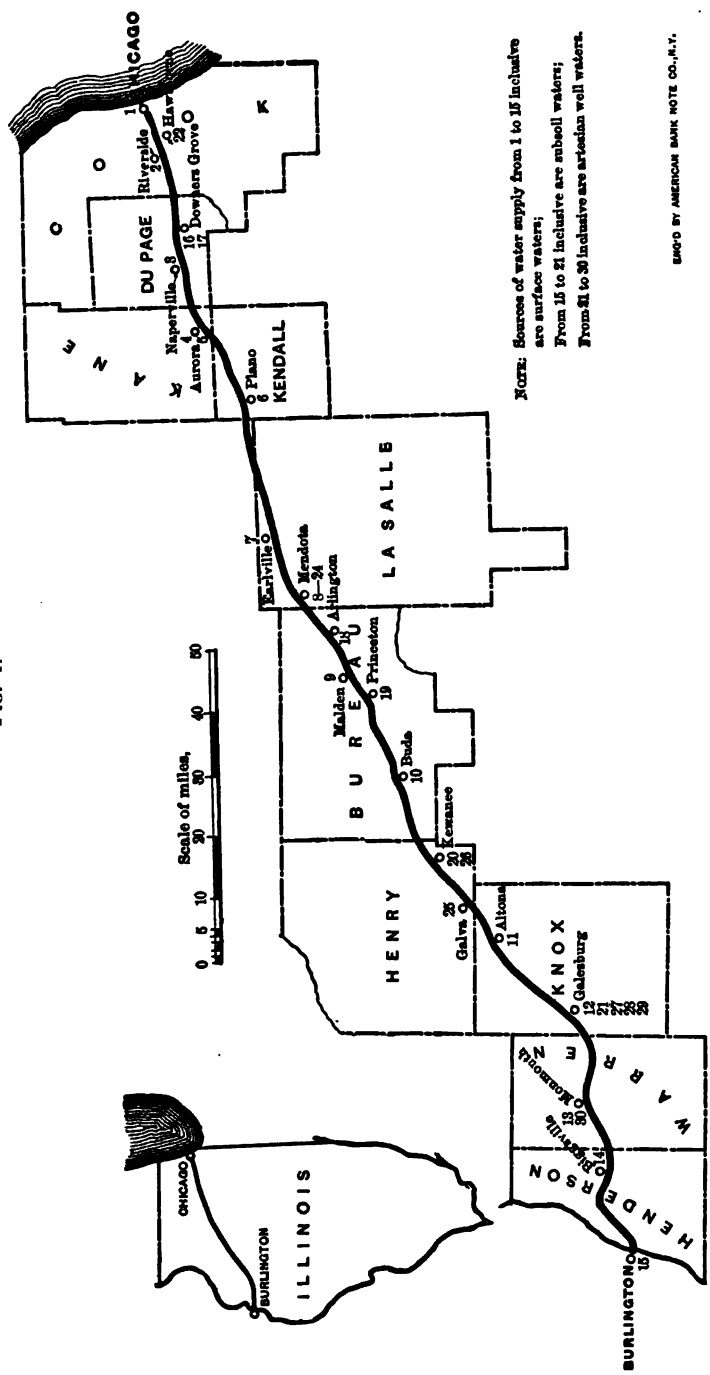


FIG. 1.

Map showing Principal Water-Stations on the Main Line of the Chicago, Burlington and Quincy Railroad from Chicago to Burlington.

mention will be made of those formations only which furnish the water-supplies herein treated.

The surface-waters are derived from the drainage of the water-shed and from springs. The surface of that portion of the State which furnishes these waters consists of glacial ridges, till-plains and loess-covered till; the water partaking of the soluble portions of these formations. No separate study has been made of the spring-waters; but they can probably be classed with the ground-waters.

The ground-waters are derived from glacial drift, which covers the older rock to a depth of from 10 to 200 feet. The drift is of very variable composition: in some portions it consists of clean gravel; in other portions, of various mixtures of clay of several varieties, gravel, boulders, or sand. It is this great variation that makes the composition of the ground-waters so uncertain.

The deep well-waters are all from the St. Peter's sandstone. This formation occurs immediately below the Trenton limestone and varies in thickness from 10 to 300 feet. It consists of fine round grains of sand, more or less cemented together. It is free from fossils, although ripple-marks have been found in the out-crop along the Rock river. There are large out-crops in Wisconsin, but in Illinois the formation is generally from 500 to 1200 feet below the surface. In Ogle and La Salle counties, however, there are out-crops of comparatively small extent. Along the main line of the C. B. & Q. R. R., there are no out-crops.

METHODS OF ANALYSIS.

The ordinary methods of analysis were used in the analyses here reported; the acids and bases being determined in separate portions, and salts calculated as follows:

The chlorine was first united with the sodium, and the excess, if any, with the magnesium. The sulphuric acid was united with the sodium (if any sodium was left after satisfying the chlorine); then with the calcium; and finally, if there was still an excess of sulphuric acid, it was united with the magnesium. The calcium, magnesium, and occasionally sodium in excess, were calculated to carbonates. The oxides consist of silica, alumina and iron oxide and are reported as such, no attempt being made to combine them with acids.

Analyses by Ellis for oxides, calcium and magnesium were made by standard methods. A separate portion was washed with water to determine the alkalies and the soluble calcium and magnesium salts; the alkalies consisting of sodium chloride and sodium sulphate were reported as alkalies, and the soluble calcium and magnesium were calculated to sulphates. This method works well with surface-waters, but cannot be used with ground- and deep well-waters.

Both schemes of calculation are open to criticism; but the first one has been found to meet our wants fully, and is an excellent method for comparing sources of water-supply.

RESULTS OF ANALYSES.

The results of analyses are compared in four tables as follows:

TABLE I.—*Surface-Waters.*—The surface-waters include flowing streams, natural or artificial ponds fed by streams or surface-drainage, and lakes.

TABLE II.—*Ground-Waters.*—These are all from shallow wells.

TABLE III.—*Deep Well-Waters.*—These are all from the St. Peter's sandstone.

TABLE IV.—This table gives the average analyses of each class of water.

TABLE V.—This table contains analyses showing sources of contamination of reservoir at Kewanee.

A comparison of the average results and a close inspection of the analyses of the different classes of waters show that the surface-waters are quite uniform. The sodium salts are never in sufficient quantity to cause trouble from foaming, and the sulphates of the alkaline earths are low, generally less than five grains per gallon, while the main bulk of the solids is composed of carbonates of calcium and magnesium. These form a comparatively soft scale, and are the least injurious of the scale-forming solids which occur in natural waters.

The ground-waters vary from fair boiler-waters to those which are positively injurious to boilers. The characteristic of these waters, as compared with surface-waters, is the comparatively large quantity of alkaline-earth sulphates which they contain. They are low in sodium salts, and do not cause trouble

TABLE I.—*Analyses of Surface-Waters from Streams, Lakes and Artificial Ponds.*

No.	Locality.	Date.	Analyst.	Source of Supply.	In Grains per U. S. Gallon of 231 Cubic Inches.									Remarks.
					Sodium Chloride.	Sodium Sulphate.	Calcium Sulphate.	Calcium Carbonate.	Magnesium Sulphate.	Magnesium Carbonate.	Oxides, Iron, Silica, etc.	Total Solids.	Incrusting Solids.	
1	Chicago.	1885	Long.	Lake Michigan.	.22	.28	.31	4.46	.....	2.20	.33	7.80	7.30	
2	Riverside.	1886	Ellis.	Des Plaines River.	1.36	.....	.77	5.24	.....	4.02	.63	12.02	10.66	
3	Naperville.	1886	Ellis.	Du Page River.	1.07	.....	.27	9.29	9.11	.....	.55	20.29	19.22	
4	Aurora.	1897	Carney.	Fox River.	1.14	.....	5.18	4.00	.....	6.50	.17	16.99	15.85	River normal.
5	Aurora.	1894	Carney.	Fox River.	1.57	.....	4.76	5.11	.....	9.27	.38	21.09	19.52	River very low.
6	Plano.	1886	Ellis.	Big Rock Creek.	1.70	.....	.84	7.75	.....	5.46	.71	15.96	14.26	
7	Earl.	1885	Tilemann.	Indian Creek.	*2.39	.....	.23	8.80	.....	3.50	1.40	16.32	13.93	
8	Mendota.	1896	Carney.	Artificial pond.	.66	.....	1.03	5.29	.....	3.59	.27	10.84	10.18	Much organic matter.
9	Malden.	1886	Ellis.	Artificial pond.	*4.38	.....	1.74	7.97	4.26	5.22	.60	24.17	19.79	
10	Buda.	1896	Carney.	Coal Creek.	.49	.....	2.82	8.86	.....	7.46	.50	20.13	19.64	
11	Altona.	1886	Ellis.	Stevens Creek.	*4.22	.....	.15	10.18	.15	7.09	1.10	22.89	18.67	
12	Galesburg.	1886	Ellis.	Artificial pond.	*2.72	.....	.16	5.14	.35	5.32	.74	14.43	11.71	
13	Monmouth.	1896	Carney.	Artificial pond.	1.21	.....	4.34	7.57	.....	6.62	.41	20.15	18.94	
14	Biggsville.	1886	Ellis.	Henderson Creek.	1.89	.....	.19	7.21	1.12	3.89	.69	14.99	13.10	
15	Burlington.	1894	Carney.	Mississippi River.	.51	.....	.97	3.68	.....	3.16	.38	8.70	8.19	Sediment 9.99 grains.

* Includes sulphates and chlorides.

TABLE II.—*Analyses of Ground-Waters from Shallow Wells.*

No.	Locality.	Date.	Analyst.	Source of Supply.	In Grains per U. S. Gallon of 231 Cubic Inches.									Remarks.
					Sodium Chloride.	Calcium Sulphate.	Calcium Carbonate.	Magnesium Chloride.	Magnesium Sulphate.	Magnesium Carbonate.	Oxides, Iron, Silica, etc.	Total Solids.	Incrusting Solids.	
16.....	Downer's Grove.	1897	Carney.	Well 16 feet deep.	.78	41.08	.....	1.48	3.34	18.63	.50	65.81	65.03	No definite data
17.....	Downer's Grove.	1895	Carney.	Water-works well.	1.60	6.15	10.46	.....	.....	10.17	.11	28.49	26.89	about these wells
18.....	Arlington.	1887	Ellis.	Well: no data.	1.76	.21	12.22	.....	2.62	6.17	1.33	24.31	22.55	further than that
19.....	Princeton.	1896	Carney.	Well 23½ feet deep.	2.95	16.70	10.27	2.35	.....	12.56	.27	45.10	42.15	their source of
20.....	Kewanee.	1887	Ellis.	Well: no data.	3.45	.32	18.11	.....	7.11	6.42	1.27	36.68	33.23	supply is glacial
21.....	Galesburg.	1887	Ellis.	Crystal Springs well.	1.62	.18	12.19	.....	.71	6.22	1.47	22.39	20.77	drift.

TABLE III.—*Analyses of Deep Waters from Artesian Wells.*

No.	Locality.	Date.	Analyst.	Source of Supply.	In Grains per U. S. Gallon of 231 Cubic Inches.										Remarks.
					Sodium Chloride.	Sodium Sulphate.	Sodium Carbonate.	Calcium Sulphate.	Calcium Carbonate.	Magnesium Sulphate.	Magnesium Carbonate.	Oxides, Iron, Silica, etc.	Total Solids.	Incrusting Solids.	
22	Hawthorne.	1888	Ellis.	Artesian well, 989 feet deep.	*14.11	.....	.....	.51	11.72	7.36	4.58	.71	38.99	24.88	In St. Peter's sandstone.
24...	Mendota.	1896	Carney.	Artesian well, 504 feet deep.	1.62	.....	.....	.....	10.23	.....	6.74	.78	19.37	17.75	" " "
25...	Galva.	1896	Carney.	City W. W. well, 1400 feet deep.	17.33	24.48	3.34	.....	5.84	.....	5.03	.11	56.13	10.98	" " "
26...	Kewanee.	1894	Carney.	City W. W. well, 1430 feet deep.	41.32	22.44	.....	.....	10.50	.....	6.53	.43	81.22	17.46	" " "
27...	Galesburg.	1896	Carney.	City W. W. well, No. 1, 1226 feet deep.	14.32	42.32	.....	9.16	6.37	.....	9.41	.03	81.61	24.97	" " "
28...	Galesburg.	1896	Carney.	City W. W. well, No. 2, 1236 feet deep.	20.17	50.91	.....	.....	8.04	.....	10.70	.61	90.43	19.35	" " "
29...	Galesburg.	1896	Carney.	Artificial Ice Co. well, 1236 feet deep.	16.52	56.73	.....	21.77	1.51	.....	11.46	.05	108.04	34.79	" " "
30...	Monmouth.	1896	Carney.	City W. W. well, 1220 feet deep.	19.75	42.11	.....	26.60	.....	.....	12.28	.36	101.10	39.24	" " "

* Includes sulphates and chlorides.

TABLE IV—*Average Analyses of the Different Classes of Waters.*

	In Grains per U. S. Gallon of 231 Cubic Inches.										
	Sodium Chloride.	Sodium Sulphate.	Sodium Carbonate.	Calcium Sulphate.	Calcium Carbonate.	Magnesium Chloride.	Magnesium Sulphate.	Magnesium Carbonate.	Oxides, Iron, Silica, etc.	Total Solids.	Incrusting Solids.
Surface-waters (streams, lakes and artificial ponds)...	1.70	.02	.....	1.50	6.71	.....	1.00	4.88	.59	16.45	14.78
Ground-waters (shallow wells).....	2.02	.....	.....	10.77	10.54	.64	2.29	10.03	.82	37.11	35.10
Deep waters (artesian wells).....	18.14	29.87	.41	7.25	6.77	.....	.92	8.34	.36	72.08	23.66

TABLE V.—*Analyses Showing the Contamination of the Kewanee Reservoir.*

Date.	Analyst.	Source of Supply.	In Grains per U. S. Gallon of 231 Cubic Inches.									Remarks.
			Sodium Chloride.	Sodium Sulphate.	Calcium Sulphate.	Calcium Carbonate.	Magnesium Sulphate.	Magnesium Carbonate.	Oxides, Iron, Silica, etc.	Total Solids.	Incrusting Solids.	
1886	Ellis.	First analysis of pond.	*5.37	.....	.50	10.15	8.20	3.18	.21	27.61	22.24	Fair boiler-water.
1894	Carney.	Last analysis of pond.	46.28	19.94	50.90	.....	19.38	9.06	.45	146.01	78.89	Very bad boiler-water.
1894	Carney.	Kewanee water-works furnishing city supply.	41.33	22.43	.....	10.50	.....	6.53	.43	81.22	17.46	These analyses show the sources of contamination.
1894	Carney.	Small stream containing city drainage.	31.41	11.99	16.44	3.25	.....	9.45	.39	72.93	29.53	
1894	Carney.	Sewer from galvanizing-works.	63.12	20.85	26.14	4.41	.....	14.88	1.06	130.46	46.49	

* Includes sulphates and chlorides.

from foaming; but they do form a bad scale. A calcium-sulphate scale is very compact and hard, and seriously interferes with the making of steam. An experiment on the effect of scale showed that a calcium-sulphate scale, about $\frac{11}{100}$ of an inch thick, caused a loss in evaporating-efficiency of over 7 per cent. Two of the waters contain magnesium chloride. This is corrosive, and renders a water unfit for boiler-use. The principal feature of the ground-waters is their entire lack of uniformity. Analysis of several ground-waters throws no light on what the next may be.

The deep well-waters are all from the St. Peter's sandstone. The characteristics of these waters, with the exception of the Mendota well, where the sandstone comes within 460 feet of the surface, are large quantities of sodium salts, both chlorides and sulphates. The incrusting solids vary greatly in quantity as well as in composition; some being very high in calcium-sulphate, and others containing no alkaline-earth sulphates whatever. These waters cannot be successfully used for boiler-purposes, on account of the foaming-tendency due to the large quantity of sodium salts.

Foaming.

Foaming is caused by interference with the free escape of steam from the water in the boiler, and manifests itself by the rising and frequent appearance of boiling of the water in the water-glass, and by water in the steam. This does not mean wet steam, as ordinarily defined, but steam that contains visible drops of water. There are three principal causes of foaming: (1) sodium salts; (2) mud or suspended matter; and (3) organic matter.

Mud and organic matter can be removed by filtration, but there is no known method of getting rid of sodium salts; and because of this fact mention will be made of the foaming caused by sodium salts only.

The sodium salts in solution increase the surface-tension and thereby prevent the free escape of steam from the water. Bubbles, formed in rapid succession, constitute a froth which fills the steam-space of the boiler and passes over with the steam. The height to which the froth will rise depends upon the rapidity with which the steam is made. This is very nicely

illustrated by blowing air into a weak soap-solution. The more rapid the introduction of the air, the larger the quantity of bubbles formed, and the higher they rise before they break.

It has been found by actual experience that locomotive-boilers will foam with from 75 to 200 grains of alkali per gallon, depending upon how much mud or suspended matter is in the boiler, while stationary boilers have been successfully run without foaming with a concentration of sodium salts amounting to 650 grains and more per gallon.

The following explanation is offered to show why a stationary boiler will run successfully with 650 grains of sodium salts per gallon, while a locomotive-boiler will foam on from one-third to one-sixth of that quantity of alkali:

A locomotive-boiler with a shell 60 inches in diameter, tubes 12 feet 4 inches long, and a 9-foot fire-box, has a heating-surface of 1689 square feet, and develops 1000 horse-power when doing maximum work. The evaporating-surface is 96.5 square feet. Therefore, 0.59 horse-power is developed per square foot of heating-surface and 10.3 horse-power per square foot of evaporating-surface.* On the other hand, a stationary boiler, with a shell 66 inches in diameter and 18 feet long, has a heating-surface of 736 square feet and develops 100 horse-power. The evaporating-surface is 84 square feet. Therefore, 0.13 horse-power is developed per square foot of heating-surface, and 1.19 horse-power per square foot of evaporating-surface.

The ratio between heating-surfaces of stationary- and locomotive-boilers per horse-power is therefore, Stationary: Locomotive = 4.36 : 1. And the ratio between the evaporating-surfaces per horse-power is Stationary: Locomotive = 8.70 : 1. In other words, a stationary-boiler has 4.36 times as much heating-surface per horse-power for making steam and 8.70 times as much surface per horse-power for the escape of the steam after it is made, as a locomotive-boiler. Since foaming is determined by the rate of formation and the escape of bubbles of steam, it will readily be seen that a stationary-boiler with 8.70 times the evaporating-surface, and 4.36 times the heating surface, per horse-power will run successfully on a quantity

* By heating-surface is meant the square feet of water exposed to the heat from the fire. By evaporating-surface is meant the square feet of surface of water not coming in contact with any portion of the boiler.

of alkali in the water that would render the locomotive-boiler useless.

The Location of Reservoirs.

In locating reservoirs for boiler-supply, great care must be taken to see that the drainage is surface-water only, and that there is no chance of contaminating waters getting into the reservoir.

The reservoir at Kewanee was abandoned on account of the contaminating effect of waters draining into it. This water was originally as good as that of our other artificial ponds, but water from an artesian well high in sodium salts, and refuse from a galvanizing-plant, have so deteriorated the supply that it is impossible to use it in a boiler. See Table V. for comparative analyses of the water.

Conclusions.

Water for boiler-purposes should be taken as much as possible from streams, lakes, or artificial ponds. The use of subsoil-water is open to objection, unless an analysis shows the water to be sufficiently good for boiler-use. Waters from deep wells in the St. Peter's sandstone will cause foaming unless very carefully handled. The large quantity of sodium salts prevents the formation of scale, and, with regular washing-out, these waters have been used with success in stationary boilers; but for locomotive-use they are a failure.

Brief Note on Rail-Specifications.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Chicago Meeting, February, 1897.)

At the Atlanta Meeting of the Institute, October, 1895, I had the honor of presenting a paper on "Specifications for Steel Rails of Heavy Sections Manufactured West of the Alleghenies,"* in which, reasoning from the experience of Western railroads with the heavier-sectioned steel rails, I called atten-

* *Trans.*, xxv., 653.

tion to the importance of the chemical composition of the steel, and urged the increase of the carbon-percentage, and the limiting of phosphorus to the lowest point reconcilable with commercial considerations—and also insisted on the value of *added* silicon.

My paper has been honored by the favorable consideration of a number of the engineers and other controlling officers of western roads; and, as a result, more than 186,000 gross tons of rails of sections heavier than 65 pounds per yard have been manufactured since that time in accordance with my views. These rails are in the tracks of several large railway-systems. In giving the above tonnage I refer only to the rails which came under my immediate supervision; that is, under the inspection of my firm. Over 69,000 tons of these weighed 80 pounds to the yard.

In some cases it was not commercially possible to obtain quite as low a percentage of phosphorus as was desired, and hence the carbon was reduced proportionally; but, as a whole, the rails could be classed as high-carbon ones. Of course, it is yet too early to form a positive opinion as to the wear of the rails; but practically all of them have been in service long enough to demonstrate their safety, and I believe that in all cases the users of the rails unite in the opinion that, so far, they promise much better wear than that previously obtained. I know that, in several instances, the showing thus made will lead to the requirement of still harder steel for this year's deliveries.

I present this short note as somewhat in the nature of a "report of progress." Moreover, in view of recent somewhat surprising developments in the American rail-trade, it is well not to lose sight of the necessity of not permitting low prices to be the only consideration. The first cost of rails is not all the story. This proposition does not require discussion. If it is necessary to pay a little more for a good rail, true economy speaks in no uncertain voice in favor of so doing. Low prices are very attractive; but subsequent economies in track maintenance, repairs to rolling stock, possibilities of high speed, and comfort of passengers must not be ignored.

The peculiarities of human nature must be remembered. They do not change very much. When articles are sold at a low price, those in direct charge of the manufacture of them will,

most likely, be expected to keep the cost of production down to a corresponding degree. This may lead to temptation to cut some corners, which would better have remained in full projection. At all events, it does not seem wise to ignore past experience in steel-rail metallurgy.

Notes on the Determination of Insoluble Phosphorus in Iron-Ores.

BY CHAS. T. MIXER AND HOWARD W. DUBOIS, PHILADELPHIA, PA.

(Chicago Meeting, February, 1897.)

ONLY within the past few years have chemists recognized the importance of the fact, that comparatively large amounts of phosphorus may occur in the siliceous residue left from the acid treatment of iron-ores. We know of one case in which an ore contained about three times as much phosphorus in the insoluble, as in the soluble, form. It was disposed of at a premium, as an exceptionally high-grade Bessemer ore containing 0.010 per cent. of soluble phosphorus, while the insoluble phosphorus brought the total amount up to 0.040 per cent.

The magnetite and specular hematite of the Lake Superior districts and the fine ores of the Mesabi range, as a rule, contain very small amounts of insoluble phosphorus. But the mining of the soft hematites and the progressive lowering of the phosphorus-limit in a strictly Bessemer ore, have combined to make the determination of insoluble phosphorus one of the routine-requirements in all analyses of Bessemer ores.

The insoluble phosphorus is understood to be that phosphorus which cannot be extracted by boiling hydrochloric acid of 1.1 specific gravity. The time given for the extraction of the soluble phosphorus will vary, of course, according to the nature of the ore. In ordinary practice the boiling is continued until the residue is white or only very slightly tinged with ferric oxide. This ordinarily takes from half an hour to an hour. The prolonged boiling of ores known to contain considerable quantities of insoluble phosphorus has failed to extract any appreciable amount of additional phosphorus. Very fine grinding and subsequent sifting through bolting-cloth has not increased the extraction. These statements represent the re-

sult of experiments carried out to test the opinions of a few chemists, who have maintained that such operations would materially increase the solubility of the phosphorus, generally supposed to be insoluble. The determination of this insoluble phosphorus in laboratories required to make a large number of analyses of ores daily, has added a considerable burden to the ordinary routine work.

The current practice in the treatment of the residue remaining from the acid solution of the ore, in order to transform the phosphorus into soluble form, is to fuse the siliceous residue with sodium carbonate in platinum crucibles, dissolve the fused mass in weak hydrochloric acid and evaporate to dryness to dehydrate the silica. The sodium phosphate is then extracted with weak hydrochloric acid and treated in the usual way for obtaining the precipitate of ammonium phospho-molybdate. This operation requires considerable time and manipulation, and involves the introduction of sodium salts, which sometimes prove unfavorable to the obtaining of a pure precipitate of ammonium phospho-molybdate.

The practice of fusing the ore direct with sodium carbonate, and thus extracting the total phosphorus, requires larger platinum crucibles, and would not be practicable where so many determinations have to be carried on simultaneously, as in the case under consideration.

Hydrofluoric acid has been used to dissolve the insoluble residue, but care has to be taken to evaporate the excess of acid employed if the solution is to be subsequently placed in glass beakers.

Since there has been such increased demand for siliceous ores to mix with the low-silica ores of the Mesabi range, chemists have been more than ever annoyed with the determination of insoluble phosphorus, as the increased amount of siliceous matter in the residue has required for fusion proportionately more sodium carbonate, heat, time and patience.

For this reason we began a series of experiments with the purpose of finding a more rapid and convenient method of determining, either the total phosphorus or the insoluble phosphorus by itself.

In the first experiment the ore was intimately mixed with less than an equal bulk of sodium carbonate, and then sub-

jected to a red heat in a platinum crucible; the idea being to convert all the phosphorus into sodium phosphate, without using sufficient sodium carbonate to make a liquid fusion. The results were encouraging. The calcined mixture of ore and sodium carbonate was readily freed from the crucible and easily broken up by the pressure of a glass rod in the beaker. The mass was then boiled, in some cases, with water alone, and, in other cases, with weak acids, and the total phosphorus was quickly extracted from many ores containing considerable amounts of insoluble phosphorus. The main objection to this method was encountered, however, when the siliceous ores were treated by it. In such cases, owing to the large amount of siliceous residue, no matter how little sodium carbonate was used, it was difficult to prevent a partial fusion, forming silicate of sodium, and thus making it hard to remove the calcined mass from the crucible.

Calcined magnesia was next tried as a base to combine with the insoluble phosphorus, and excellent results were obtained; no trouble being experienced with the fusion of the large siliceous residues. It was, however, somewhat surprising that the magnesia acted so readily in extracting the phosphorus.

In the experiments next made, the ore was calcined without the admixture of any base, and after this operation was treated in the usual way, with hydrochloric acid, when it was found that the total phosphorus had been extracted. This treatment was successful with most of the ores tried, but had the disadvantage of rendering the ferric oxide less readily soluble, and thus increasing the time required for the subsequent solution.

Applying this idea of simple ignition to the insoluble residues only, it was found that all the insoluble phosphorus could be thus converted into the soluble form, and a very simple practical method was thus established.

Very recently our attention has been called to the fact that Mr. Lychenheim* and Mr. Norris† had come to the same conclusion in the determination of phosphorus in coal and coke—namely, that fusion can be dispensed with and simple ignition substituted. Mr. Lychenheim has also informed the authors that he has found it to be perfectly satisfactory in the case of ores.

* *Trans.*, xxiv., 66. † *Ibid.*, xxiv., 862.

The details of our practice are as follows: About 1½ grammes of ore is dissolved in a No. 3 beaker with 25 c.c. of hydrochloric acid, 1.1 specific gravity. When the ore is dissolved, the excess of acid is evaporated until the solution begins to assume a syrupy consistency. It is then diluted with water and filtered into an Erlenmeyer flask, and the filter-paper and residue are placed in a platinum-crucible and ignited. When the paper is burned off the residue is broken up with a platinum-rod and ignited at a red heat for a couple of minutes, when it is removed and placed in a beaker for solution. Water is added, together with a few drops of hydrochloric or nitric acid, and the solution is brought to a gentle boil for about five minutes. It is then filtered into the flask containing the soluble phosphorus (or into another flask, in case it is to be determined separately*), neutralized with ammonia and precipitated as ammonium phospho-molybdate. The latter precipitate is titrated according to Handy's modification of the sodium hydroxide method.

This method having been found by many tests to give perfectly reliable results, has been in use in our laboratory for more than a year. The table on the following page exhibits a few of the many analyses made by us to satisfy ourselves as to the accuracy of the method.

In order to obtain some general idea of the nature of the base, which was combined with the phosphorus in the insoluble form, we made the following partial analysis of the insoluble residue from the hydrochloric acid treatment.

After drying the residue at 100° C., it was subjected to ignition in a platinum crucible for five minutes:

	Per cent.
Loss on ignition,	5.05
Loss due to extraction by acid (*a*),	16.25

The residue from the above treatment contained, in percentages of the original residue:

	Per cent.
SiO_2,	74.25
Al_2O_3,	2.70
CaO,	1.37

* It has been found that it is better to determine separately the soluble and insoluble phosphorus. Otherwise a too dilute solution is liable to be obtained for the precipitation.

Table of Phosphorus-Determinations.

METHOD A.—Solution of ore and fusion of residue with sodium carbonate (old standard method).

METHOD B.—Solution of ore and ignition of residue without flux (proposed method).

METHOD C.—Ignition of ore without flux, and subsequent solution, determining total phosphorus.

Name of Ore.	Soluble Phosphorus.	Insoluble Phosphorus.	Total Phosphorus.	Method.
	Per cent.	Per cent.	Per cent.	
1. Winthrop (Nos. 1, 2, 3 and 4 are siliceous ores.)	0.051	0.008	0.059	A.
"	0.051	0.008	0.059	B.
2. "	0.039	0.008	0.047	A.
"	0.039	0.008	0.047	B.
3. "	0.054	0.014	0.068	A.
"	0.052	0.015	0.067	B.
"	.....	.....	0.067	C.
4. "	0.051	0.008	0.059	A.
"	.....	.....	0.058	C.
5. Cambria	0.052	0.004	0.056	A.
"	0.053	0.003	0.056	B.
6. Lillie	0.065	0.006	0.071	A.
"	0.065	0.007	0.072	B.
7. Lake Superior	0.021	0.006	0.027	A.
"	0.020	0.006	0.026	B.
8. "	0.112	0.022	0.134	A.
"	.....	.....	0.135	C.
9. Salisbury	0.058	0.009	0.067	A.
"	0.058	0.010	0.068	B.
10. "	0.028	0.019	0.047	A.
"	0.028	0.019	0.047	B.
11. "	0.046	0.006	0.052	A.
"	0.046	0.006	0.052	B.
Cleveland Hematite	0.022	0.015	0.037	A.
" "	0.022	0.016	0.038	B.
" "	0.021	0.016	0.037	B.
" "	.....	.....	0.038	C.

The solution (marked *a*) from the ignited residue contained, in percentages of the original residue:

	Per cent.
Al_2O_3,	9.55
CaO,	0.92
P_2O_5	4.10

From the latter analysis it would appear that the greater part, if not all, of the phosphorus was combined with the alumina.

The conversion of the insoluble phosphorus into the soluble form by simple ignition is a matter of some theoretical interest. The formation of a more soluble phosphate under such conditions certainly does not appear probable. Possibly

the method proposed by Berzelius* for the decomposition of phosphates, by means of silica and sodium carbonate may involve a reaction somewhat similar to that which we have here under consideration.

The Geology of the Magnetites near Port Henry, N. Y., and Especially those of Mineville.

BY J. F. KEMP, NEW YORK.

(Chicago Meeting, February, 1897.)

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* Fresenius, 6th German edition, I., 416.

Introductory Note.

Having spent a portion of nearly every summer, since 1887, in systematic geological work in the Adirondacks, involving special study of the iron-ore-bodies in and near Mineville, N. Y., the writer prepared, in 1893, for the Director of the N. Y. State Museum, a report on the geology of Moriah and Westport townships, accompanied by sections of the mines, which (complete data not being then available) had been prepared after correspondence, in 1892 and 1893, with Mr. S. B. McKee, the engineer of the two companies operating the mines, and with Mr. E. B. Durham, his assistant at that time. During this work it became evident that more numerous sections would be of great interest, and would throw much light on the geology of the ore-bodies; and in 1894 Mr. McKee collected the necessary data, and the writer spent September of that year in assisting him and studying, in greater detail than before, the geological relations already recognized in their main features. This work the writer has reviewed and verified during the summer of 1896, while in the field for the U. S. Geological Survey; and, with the approval of the director, the notes of this final study have been utilized in the present paper. It is but fair to say, however, that they only serve to confirm the conclusions reached by the previous investigations.

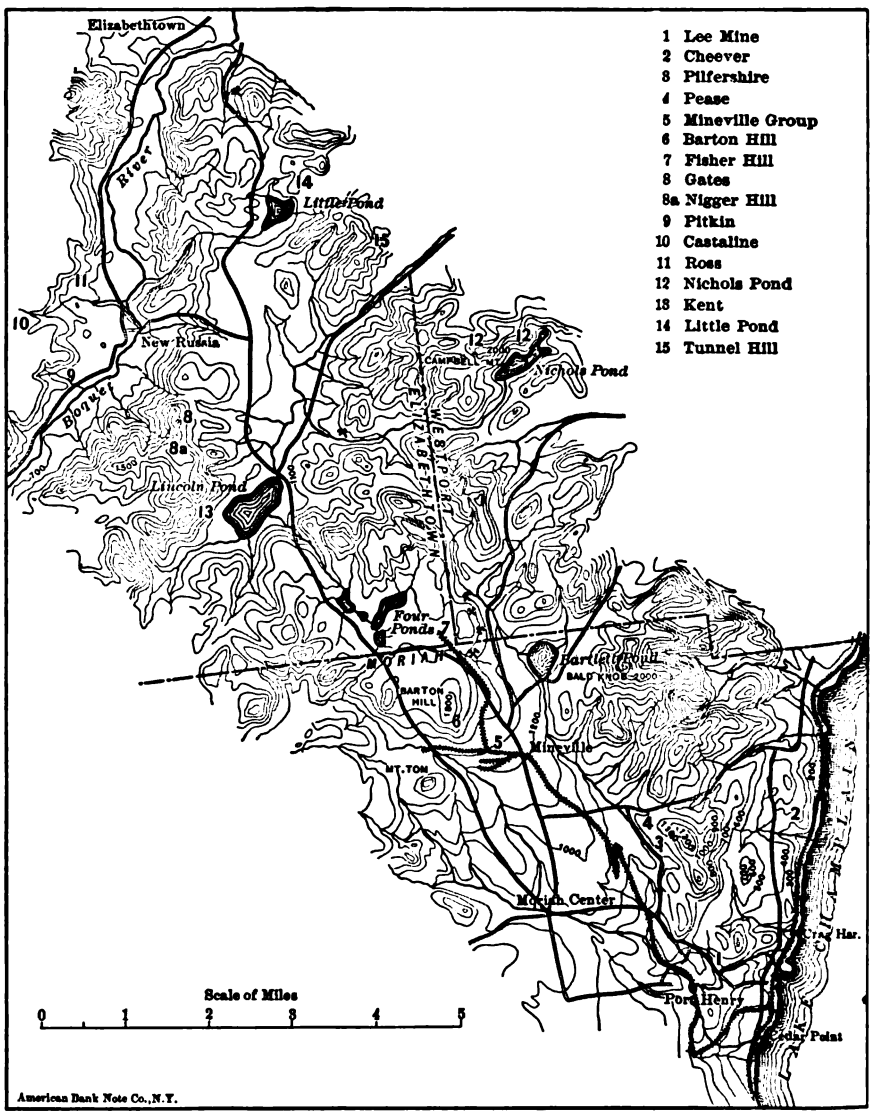
If we except the great hills of iron-ore at Cornwall, Pa., and perhaps the deposits of titaniferous iron-ore at Lake Sanford, N. Y., the Mineville ore-bodies are the largest single group hitherto developed east of Lake Superior. Occurring, as they do, on the contact between gabbro and gneiss, their geological relations differ from those of other magnetites further south, so far as the latter have been described; and they may, therefore, serve to shed some additional and special light upon the very obscure question of the geological history of this type of ore-body.

The writer takes this opportunity to acknowledge his indebtedness to the officers of Witherbee, Sherman & Co. and the Port Henry Iron Ore Co., and especially to Mr. S. B. McKee, the engineer of both companies, for abundant courtesy and assistance in the work.

General Topography.

The accompanying topographical map (Plate I.) is based on the Port Henry and Elizabethtown sheets of the U. S. Geologi-

Plate I



Map indicating the Topography and the Iron-Mines near Port Henry, N. Y.

cal Survey. Only the 100 feet contours are reproduced. From Port Henry, on Lake Champlain, a marked valley gradually

ascends northwest until, at Mineville, it is 1100 or 1200 feet above the level of the lake, the latter being about 100 feet above tide.* South of Port Henry a high ridge, not shown on the map, intervenes between it and Crown Point, which occupies the next valley running back from the lake. Abrupt ridges, with narrow passes, bound Moriah township on the west, and on the north high ridges intervene between Mineville and the valley of the Boquet river in Elizabethtown. Just north of Port Henry, and not far from the lake-shore, a great ridge begins which, gradually increasing in height and width, culminates in Bald Knob, over 2000 feet above tide. Still further north, along the Elizabethtown-Westport line, the country is extremely rugged, presenting high knobs and deep narrow valleys, both almost uninhabited.

These general topographical features are well brought out by the contours in Plate I.; but it may be added that in the Moriah valley itself minor hills arise, of which Barton hill, northwest of Mineville, is of special interest in connection with the ore-deposits.

General Distribution of the Mines.

The numbers and the legend on Plate I. indicate the distribution of the ore-bodies thus far discovered. The Lee and the Cheever lie on the east of the Bald Knob ridge, but both dip to the west, as do all the ore-bodies of the region. Emmons records also a Crag Harbor vein, just north of Crag Harbor, but this has not received much attention. It is said to be very near the lake-shore, and, if one may base an inference on the early experience had with its ores in the furnace, it must be titaniferous—a composition indicated, moreover, by the character of the neighboring rocks. On the west side of the Bald Knob ridge, and nearly due west of the Cheever mine, are two ore-beds, the Pilfershire and the Pease. Two or three miles across the valley to the northwest lie the Mineville and Barton Hill groups, which are much the largest of the region. Just across the township-line, in Elizabethtown, are the Fisher Hill and Burt Lot mines. Just east of Fisher hill is the O'Neill shaft, now abandoned, but formerly an important producer,

*It is given as 101 feet in the recent topographical maps of the U. S. Geological Survey.

while northeast of the O'Neill is the Cook shaft, located on the eastern side of a considerable intervening hill, but having the same character and history. On the northeast shoulder of the same hill is the Humbug mine, the shaft of which pierced a titaniferous ore-body. It is indicated on the map by crossed hammers only. Across a little valley still further north is a small abandoned mine called the Sherman, only shown by crossed hammers.

Northwest, the surface drops rapidly from 1600 to 1100 feet above tide (at Lincoln pond), and then rises again to 1600 feet, in the ridge containing the Gates and neighboring mines, now abandoned. Beyond these it descends about a mile to the valley of the Boquet river, and, in the edge of that valley, on the slopes of the hills that rise rapidly to form the noble peak of Giant mountain and its outliers, there is a series of ore-bodies that have never been much worked. About 5 miles a little to the east of north from Mineville, and high up on the slopes of Campbell mountain, 1900 feet above tide, are the abandoned Nichols Pond mines. The mine is where the western 12 is located on the map. The eastern 12 is the mill.

All of the ore-bodies above cited, except the Crag Harbor and the Humbug, are non-titaniferous, and, in some instances, of Bessemer grade. In addition to these, however, there are others of characteristic geological relations, which differ from the preceding. Of this type are the following: the Kent mine, just north of Lincoln pond; the Little Pond mines, just northeast of Little pond; and the Odell and Tunnel mountain openings, all in the town of Elizabethtown. They have essentially the same geological relations as the Split Rock mines on the shores of Lake Champlain. They form bodies, often of great size, and of shape not yet determined, in the midst of massive green or black gabbros, with which they form essentially a geological unit. The ore is clearly a phase of the gabbro magma, that is almost entirely titaniferous magnetite. Dark green plagioclase crystals are enveloped in the ore, and were clearly involved in it when it crystallized. The ores are, therefore, excessively basic segregations in a fused and cooling magma. The titaniferous ore-bodies are reserved for a future paper.

General Geology.

The area covered by the map, Plate I., is made up in largest part of a series of gneisses, with which are associated white crystalline limestones, ophicalcites, hornblende-schists, and, to a minor degree, other schists. These are penetrated by great intrusions of dark, massive gabbro, which itself has assumed in places strong gneissoid structure. In the southwestern portion of Moriah, in the western and northern parts of Elizabethtown and in Westport, occur the Norian rocks, consisting almost entirely of labradorite feldspar, and so characteristic of the Adirondacks. They are best called anorthosites, as is customary in Canada; the word meaning a rock composed of triclinic (*i. e.* lime-soda) feldspar, from the collective French name, *anorthose*, for this feldspar. They are most intimately associated with the true gabbros; and both are regarded as belonging to the same general series of intrusions, although various exposures in Elizabethtown prove that the gabbros are later than the anorthosites. The anorthosites often become fairly rich in pyroxenic minerals, and approximate the gabbros proper; but they are never as dark and basic as the typical cases of the latter, and, as a rule, there is no great difficulty in distinguishing them in the field. The geological relations of these two rocks to each other, and of both to the acidic gneisses and crystalline limestones, have been rendered very obscure by the extensive dynamic metamorphism to which all have been subjected. This seems to have been in the nature of irresistible pressure, accompanied, as a rule, with shearing stresses, so that the minerals have been crushed and drawn out into augen-gneiss, or even into ordinary gneisses. Rocks of highly obscure history are thus presented; and what were probably originally gabbros and pyroxenic anorthosites have passed into hornblende gneisses of varying basicity. It is possible that hornblendic gneisses are present, which have not been developed in this way; but, for many exposures, the process can be demonstrated by finding passages into thoroughly massive outcrops of typical gabbro, or by the presence in the gneisses of large nuclei or *Augen* of blue labradorite. These questions of metamorphism and areal geology cannot be discussed in full here, as they have but a minor bearing on the relations of the ore-deposits, and are too complex for treatment in a subordinate way. The citations

below will give access to work already done; and the field-work of the writer, now in progress, in the preparation of an atlas-sheet for the United States Geological Survey, will in time set them forth in detail.*

The strike of the gneisses, when it can be detected, is quite variable, ranging from northeast to northwest. It is seldom east and west, but approximates a northerly trend. The gabbro intrusions are, however, so numerous, and cover such a wide area, that any original strike in the older gneisses is pretty effectually destroyed, and, except as indicated by the general trend of the crystalline limestones, is an uncertain quantity. Its geological significance is involved in the question, to what extent the older acidic gneisses are to be considered metamorphosed sediments; but, in the present state of our knowledge, no well-informed person would wish to commit himself on this point. The crystalline limestones have much graphite, and are occasionally associated with minor schists and apparently with quartzites, which give good ground for regarding them as sedimentary in origin. The acidic gneisses, so far as can yet be declared, may have been granites or quartz-diorites.

The stratigraphic succession is quite certainly as follows: The quartzose gneisses and crystalline limestones, with perhaps some more basic gneisses, are the oldest rocks. The limestones lie above some gneisses, and appear to favor the upper part of the series, but excellent exposures occur where they are certainly below other members. The anorthosite intrusions followed in time, and furnished the materials of the highest peaks in the Adirondacks, as well as for various outlying areas and intruded sheets in the gneisses just referred to. Gabbro outbreaks fol-

* J F. Kemp, "The Geology of Moriah and Westport Townships, Essex Co., N. Y.," *Bulletin New York State Museum*, 1895, iii, 325. "Preliminary Report on the Geology of Essex Co., N. Y.," *Report of N. Y. State Geologist for* 1893, pp. 444 and following. "Gabbros on the Western Shore of Lake Champlain," *Bulletin of the Geological Society of America*, v., 213–224, 1894. "Crystalline Limestones, Ophicalcites, and Associated Schists of the Eastern Adirondacks," *Idem*, vi., 241–262, 1895. "Illustrations of the Dynamic Metamorphism of Anorthosites and Related Rock in the Adirondacks" (abstract), *Idem*, vii., 488, 1895.

For the Western Adirondacks see: C. H. Smyth, Jr., "Geological Reconnaissance in the Vicinity of Gouverneur, N. Y.," *Trans. N. Y. Acad. Sciences*, xii., 97, 1893. "Petrography of the Gneisses of the Town of Gouverneur, N. Y.," *Idem*, xii., 203, 1893. "Crystalline Limestones and Associated Rocks of the Northwest Adirondack Region," *Bulletin Geol. Soc. America*, vi., 263, 1895.

lowed, which cut the anorthosites as dikes, and formed extensive intruded knobs, bosses, and sheets in the gneisses and limestones. Dynamic metamorphism followed, imparting a foliated structure to all these rocks; and after this they experienced protracted erosion before the Cambrian times opened. The main features of their present topography were probably outlined in this period of erosion; and presumably they were heavily faulted. In the latest Cambrian time, represented by the Potsdam sandstone, the ocean encroached on the Adirondack areas so as to deposit in the valleys much Potsdam sandstone; and later the Calciferous, Chazy, and Trenton limestones, and other minor forms of sediments, and finally the Utica shales, were laid down.* No more recent sediments, other than the Glacial and post-Glacial gravels, sands, and clays have been found. It should be added that the early Cambrian strata of the Georgian series are limited to Vermont, and, so far as is now known, do not cross Lake Champlain at any point in the Adirondack region.

As a minor feature in the geology of the region, it should be mentioned also that trap dikes, usually of small width, are very common. Some of them, as shown by exposures along the lake-shore, are later than the Utica shales, which they cut; but how much later it is impossible to state. They are associated with trachytes or porphyries, which in some instances cut the traps, and are therefore subsequent to them. Some of these trap-dikes may be present in the interior also; but Prof. H. P. Cushing has shown that there is little doubt of the existence of a series there of pre-Potsdam date, but following the general metamorphism.† The writer has found also, in the interior region, boulders showing one dike cutting another, and the later one evidently chilled by the earlier, which must have been cold when the later intrusion occurred. Almost all the mines of any considerable size exhibit these dikes.

Moraines and modified drift are widespread throughout the district, and often mask the outcrops. Lake-bottoms are evi-

* See a paper by the writer on "Physiography of the Eastern Adirondacks in the Cambrian and Ordovician Periods," in *Bulletin of the Geological Society of America*, vol. viii., p. 408.

† H. P. Cushing, "On the Existence of Pre-Cambrian and Post-Ordovician Trap-Dikes in the Adirondacks," *Trans. N. Y. Acad. Sci.*, xv., 248, 1896.

dent in most of the enclosed valleys, that of Elizabethtown itself being a fine example. Deltas at several different levels appear where each stream comes in.

The Non-Titaniferous Ore-Bodies.

The Lee Mine.—This is situated a short distance west of Port Henry, in gneiss immediately west of a belt of ophicalcite and limestone, and bounded, further west, by a belt of the same rocks. There is excellent ground for thinking that a fault runs north and south just east of the mine, so that the limestones may be a repetition of the same series. If so, the relations of the ore suggest those of the Cheever mine, to which attention will next be given, although no exposures of gabbro were visible under the Lee ore-body. Outcrops, however, are few. The strike is W. of N., and the dip of the skipway is 19° W. Mr. Putnam, in the *Tenth Census*, vol. xv., p. 115, gives a plan and sections of this mine. He traced the outcrop around to the south, at the time of his visit, and showed that the strike changes to northeast, and that the dip grows steeper. The mine was then idle, and has been so ever since. It is now entirely dismantled and is full of water. A breast of over 9 feet of a pyritous magnetite is reported by Putnam, who says, besides, that the ore is cut off on the north by a trap-dike, and presumably by a fault also. The hanging-wall is shown by thin section to be a granitic gneiss, the dark silicate being biotite.

The mine had a short life, the sulphur in the ore proving a drawback. A large pile of the ore, however, that had remained on the dump for some years, so that the sulphur had been principally weathered out of it, was found salable and was shipped away for furnace mixtures. Mr. Putnam took two samples, one from a lot of 2500 tons from the north opening, and another from a lot of 1500 tons from the south opening, which gave respectively:

	I.	II.
Fe, ...	45.01	44.38
P, ...	0.047	0.04

The sulphur was not determined. The analyses show that the ores are low in iron for this district, but the phosphorus is also low.

The Cheever Mine.—This mine is situated about two miles north of Port Henry, and is one of the most interesting of the district. Its geological section, as regards the presence of gabbro or gabbro-gneiss in or near the foot-wall, is strongly suggestive of the Mineville relations. A little brook has worn out a gulch from the highway at a point south of the ore-body, for about half a mile to the lake. This flows over crystalline limestone and hornblende-schists, so far as the outcrops indicate. At the lake-shore a fine ledge of limestone is seen back of the old docks, where the brook comes in. It continues northward along the shore, across a jutting point, and gives way to a greatly brecciated quartzose gneiss beyond a small cove. This again, further north, is replaced by a great intrusion of gabbro more or less gneissoid, that underlies the ore, although not forming its immediate foot-wall. Passing westward over the gabbro, up a steep hillside and over a ridge, about 300 feet above the lake, one descends a little and meets the outcrop of the ore. The gabbro becomes somewhat foliated near the ore, and at about 50 feet from it gives place to gneiss. Over the ore is about 150 feet of gray quartzose gneiss, sometimes with pyroxene, again with hornblende. This is followed by black hornblendic schists, white crystalline, graphitic limestone and ophicalcite. The ore at first dips steeply to the west, but soon flattens, and after being heaved by a number of small normal faults, several of which contain trap-dikes, it is cut off by a fault at the west end. The outlines of the ore-body in Fig. 1 are from Mr. Putnam's paper, as the mine was working at the time of his visit, so that sections could be plotted. It has been idle and full of water during all of the writer's visits. Into this outline the writer has introduced the geology.

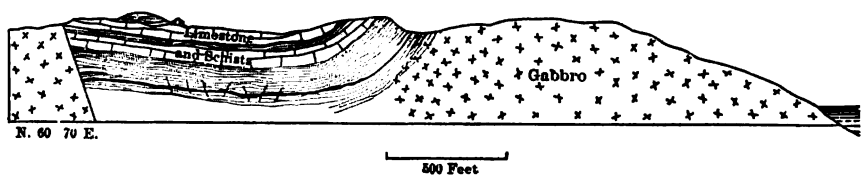
The composition of the Cheever ore is much like that of the Mineville group, as may be seen from subsequent comparisons. The following analyses were made by the chemists of the *Tenth Census* from samples taken by B. T. Putnam, underground in the first four, and from stock-piles in the last two. (*Tenth Census*, vol. xv., p. 114.)

	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Fe, .	65.33	63.5	63.86	64.42	64.77	63.08
P, .	0.643	0.603	0.689	0.452	0.673	0.573
TiO_2	Present.	Absent.	Present.	Present.	Present.	Present.

The analyses show the ore to be of high grade and of very uniform quality.

The Pilfershire and Pease Pits.—These were based upon two lenses of ore on the west side of the Bald Peak ridge and just in the edge of the valley of Moriah. For many years neither of them has been worked; they have been entirely dismantled and were full of water during the writer's visits. They strike to the northwest (N. 20 W. by compass), and dip about 50° to the west. To judge from the open pit, there must have been a breast of 10 feet, more or less, of ore. The wall-rock, now accessible, is a pyroxene-plagioclase gneiss. The pits are half a mile or less apart, and appear to be on the same line of strike and at the same horizon in the foliation of the gneiss.

Fig. 1.



Cross-section of the Cheever Mine. From *Bull. N. Y. S. Museum*, Vol. III., p. 347, Fig. 5.

The Mineville and Barton Hill Groups.

The central valley of Moriah township rises in a gradual incline from Lake Champlain, at Port Henry, northwest to Mineville, which at the large mines is situated at an altitude of 1300 feet above tide. The valley at this point is then broken by Barton and Belfry hills, around which it forks, and soon reaches the divides that bound the area of southern drainage. All the mines are, however, south of the divide, except the outlying ones at the Burt lot.

The geological structure of the valley is much obscured by the mantle of sand and drift that covers all the lower ground. Even around the mines of the lower series this is true, and except at the actual pits there is but slight opportunity to note the character of the wall-rock. As one ascends toward Barton hill, however, exposures become much more abundant, and it is possible to map the rocks in detail and to work out some very interesting relations. Such indications as are available go to show that the foot and hanging in the large workings

correspond in general character to those of the Barton hill series, although apparently not stratigraphic equivalents.

The ore-bodies form two groups contrasted in size and in the character of the ore. The lower one (both topographically and stratigraphically) is the larger producer, although it extends over a much more limited outcrop. Its ore-bodies are extraordinarily thick and have furnished vast amounts of iron. It will hereafter be spoken of as the Mineville group. The higher group is on Barton hill, and extends in an almost unbroken outcrop for about one mile, or, if (as is justifiable) the Fisher hill and Burt lot mines be considered in the same series, for over a mile and a half. The geological relations and structure of the upper group are much more intelligible than those of the Mineville group. The outcrops of the latter are situated between the 1280 and the 1340 contours, while the Barton hill outcrop runs uphill from 1300 to 1760, as is seen on the accompanying map (Plate X., facing page 202).

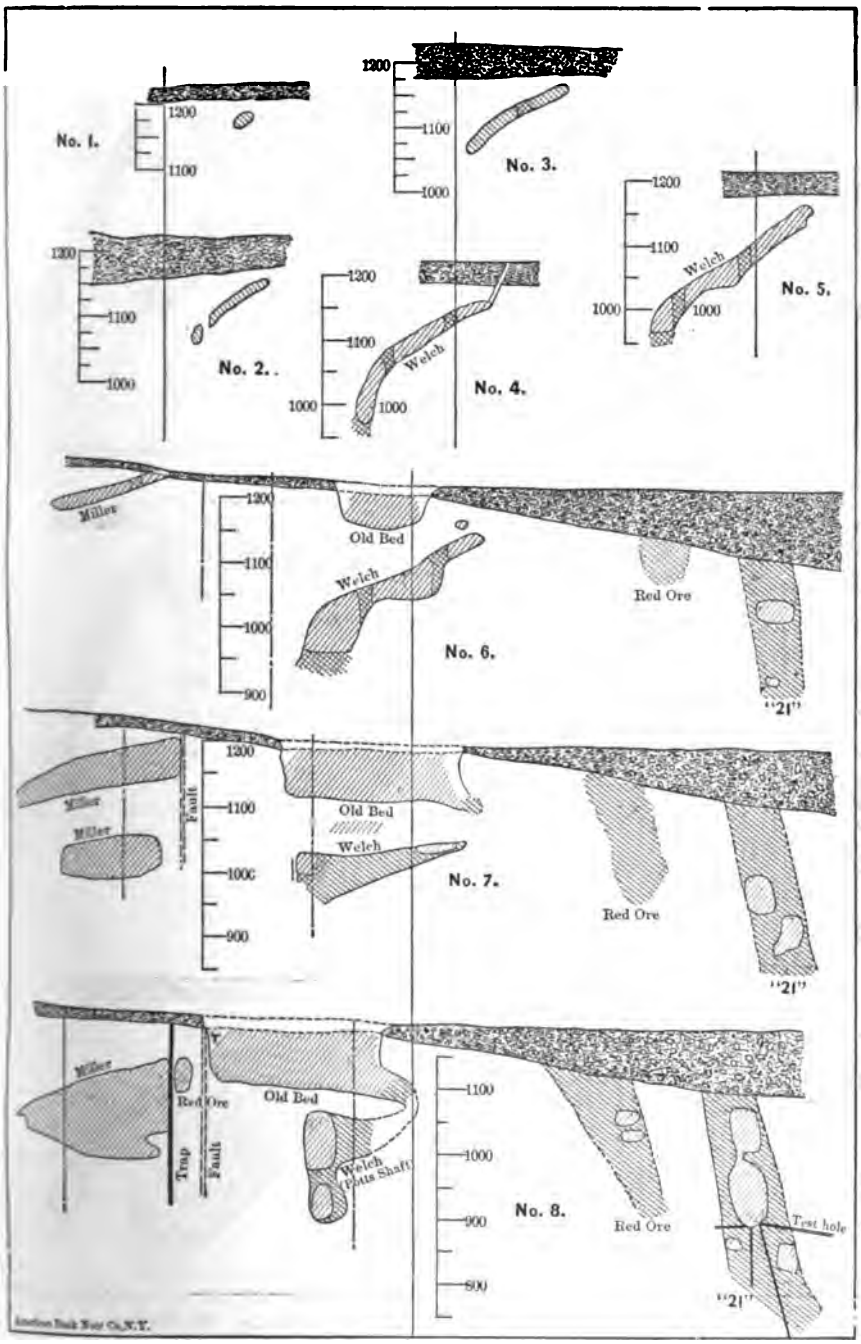
The Mineville Group.—The Mineville group embraces several huge ore-bodies, known as the Miller pit, the Welch shaft, Old Bed and Mine 21, with its extensions, the Tefft shaft, the Bonanza and the Joker. The map, Plate II., indicates their locations and areal relations. The property line that passes between the Tefft shaft and Mine 21 is a true north and south line. Mine 21 outcrops at the most southern and lowest point, and has its axial strike in a direction a little east of north. It dips to the southeast and pitches to the southwest, but it has a very complex shape, as is shown by its forking and splitting. A short distance to the north is the Old Bed, earlier known as the Sanford or "23," the numbers in all cases being the original numbers of the lots. Old Bed, with its apparent extension, the Welch shaft bed, likewise has an axial strike east of north, and a pitch to the southwest, but its dip, considered at right angles to the strike, is rather flat, and at its northern end is northwest. The Miller pit lies still further to the northwest, and has the usual northeast axial strike, with a southwest pitch and a marked dip to the northwest. It therefore is evident that, collectively speaking, the ore-bodies are a series of huge pods with their long axes in a parallel northeast direction, with a general pitch to the southwest, but with dips that on the southeast of the center line of

sections are southeast, and on the northeast of it, northwest. Parallelism to this axial alignment and pitch will be shown for the Barton hill group, but the dip is in all cases northwest. There is some faulting evident in the "21" group, and great complexity of form, as will now be followed out with the cross-section. The map, Plate II., will serve to show the locations of the several mines, as well as of the sections.

The Miller Pit.—The Miller pit is first encountered in Section 6, Plate III. In this section it is about 15 feet thick and dips at a low angle to the northwest. It pinches out at about 150 feet down. In Section 7 it has become much thicker, being over 50 feet, and has also extended to a greater distance down the dip. It is cut off by a fault on the southeast, which fault is seen in the next six sections. A lower lying portion has also come in about 75 feet below, which pinches out before Section 6 is reached. Both these join before Section 8 crosses them, and together form a magnificent cross-section nearly 150 feet thick. A trap-dike here forms the southeastern wall, beyond which, so far as excavations have indicated, only the high phosphorus ore, which, from its abundant red apatite, is called "red ore," is met. This forms a long, narrow chute that extends through eight sections. The dike apparently fills a fault-crack, for it is improbable that the ore would come to so abrupt a termination. In Section 9 the main ore again develops a split caused by the entrance of a horizontal horse. A long, narrow offshoot runs in under the horse for about 100 feet. Both parts of the ore show the dike at the end of the southeastern workings. The red ore in two chutes appears beyond the dike in the upper workings. In Section 10 the horse that entered in Section 9 has extended entirely across the pod, and has split the ore into two parts. The old upper portion that has up to this time been the main part of the mine has now dwindled, and before Section 11 is reached it pinches entirely out. The lower portion now assumes the principal rôle, and as a broad but thin pod, of the "Lima-bean" type, it extends through the next eight sections, until beyond No. 18 it pinches entirely out. In No. 16 it has split into right and left portions, with one a little below the other.

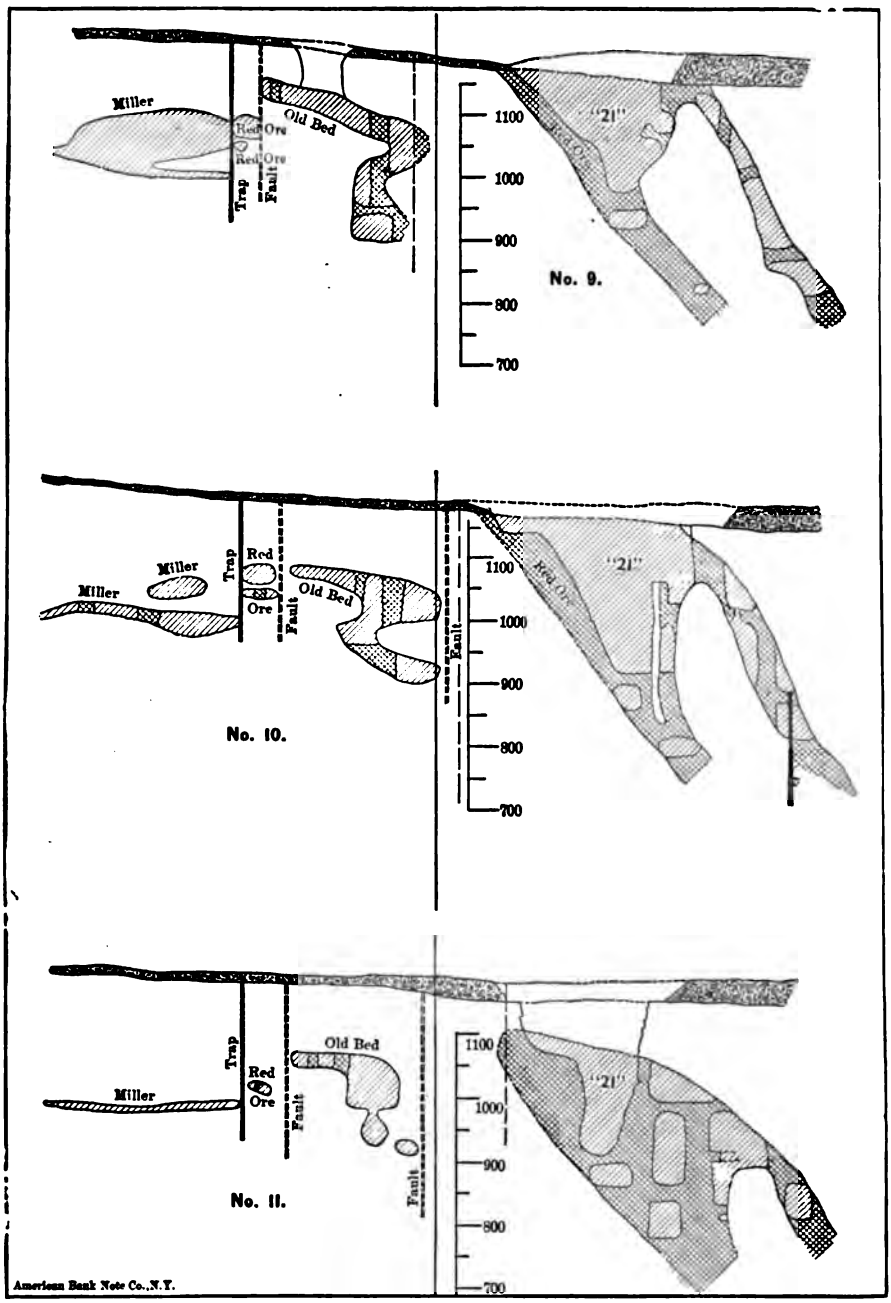
The Miller ore-body is the simplest of the Mineville group. It forks as they all do, but it lies nearly flat in its lower work-

Plate III



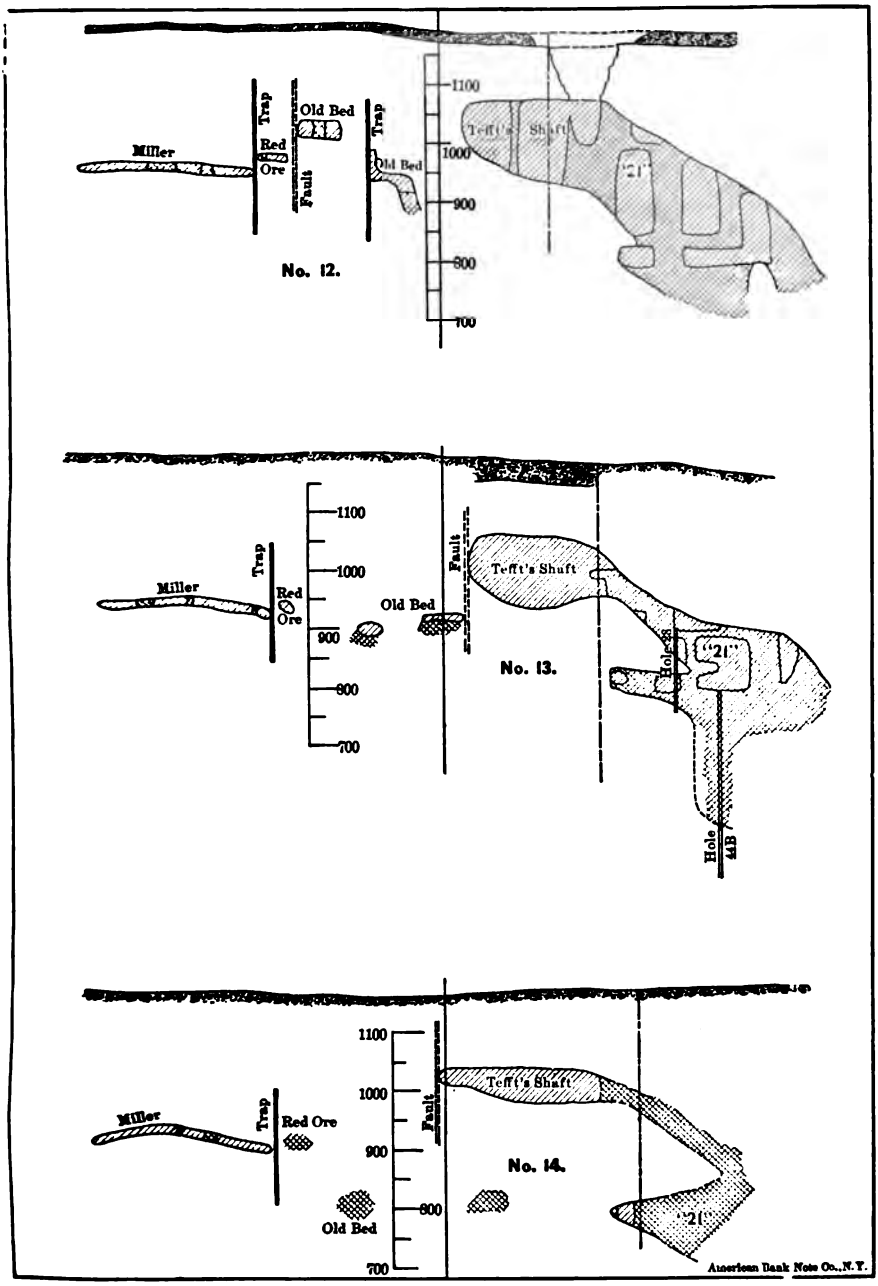
Sections 1 to 8, inclusive, of the Mineville Group. See Plate II.

Plate IV



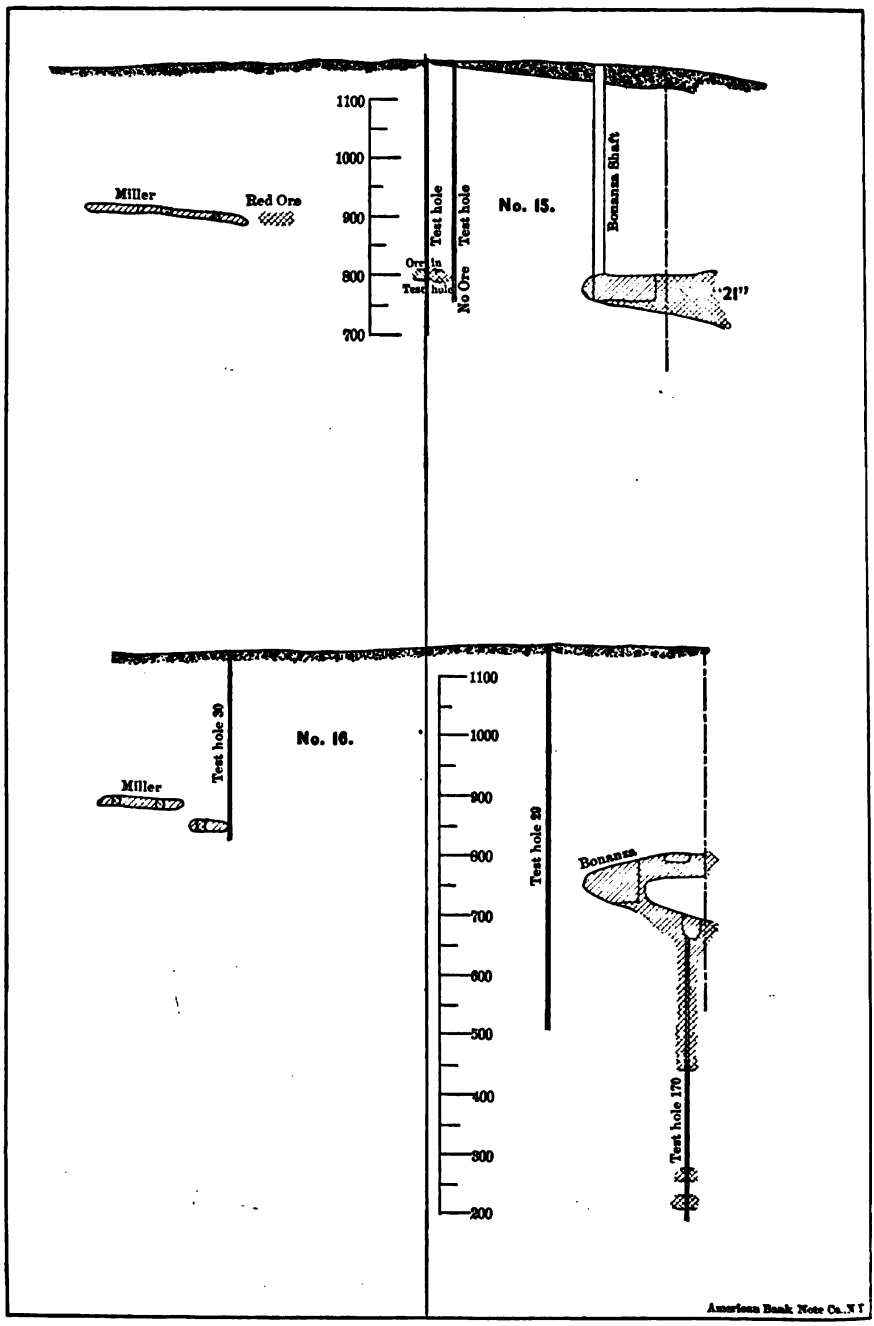
Sections 9 to 11, inclusive, of the Mineville Group. See Plate II.

Plate V



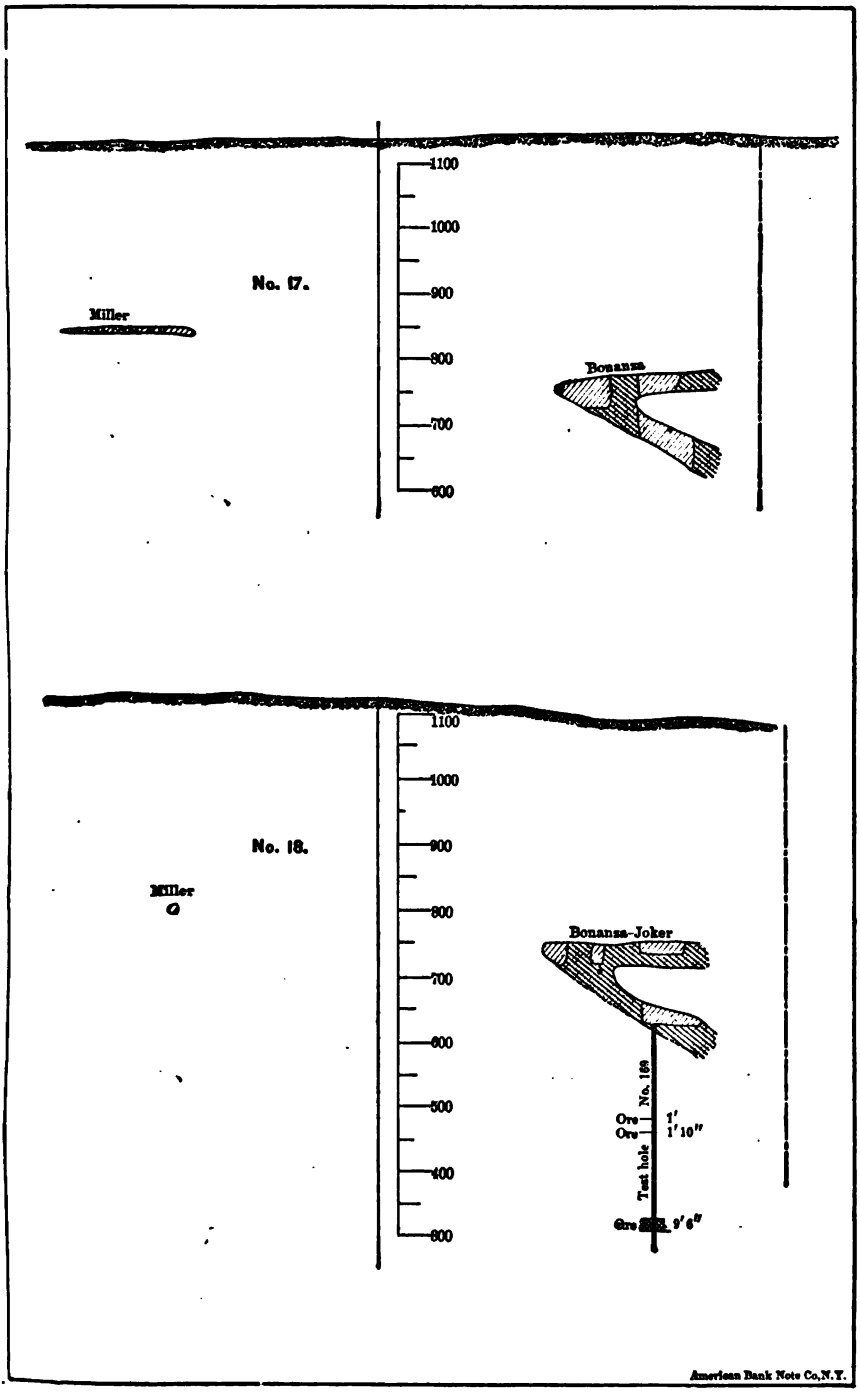
Sections 12 to 14, inclusive, of the Mineville Group. See Plate II.

Plate VI



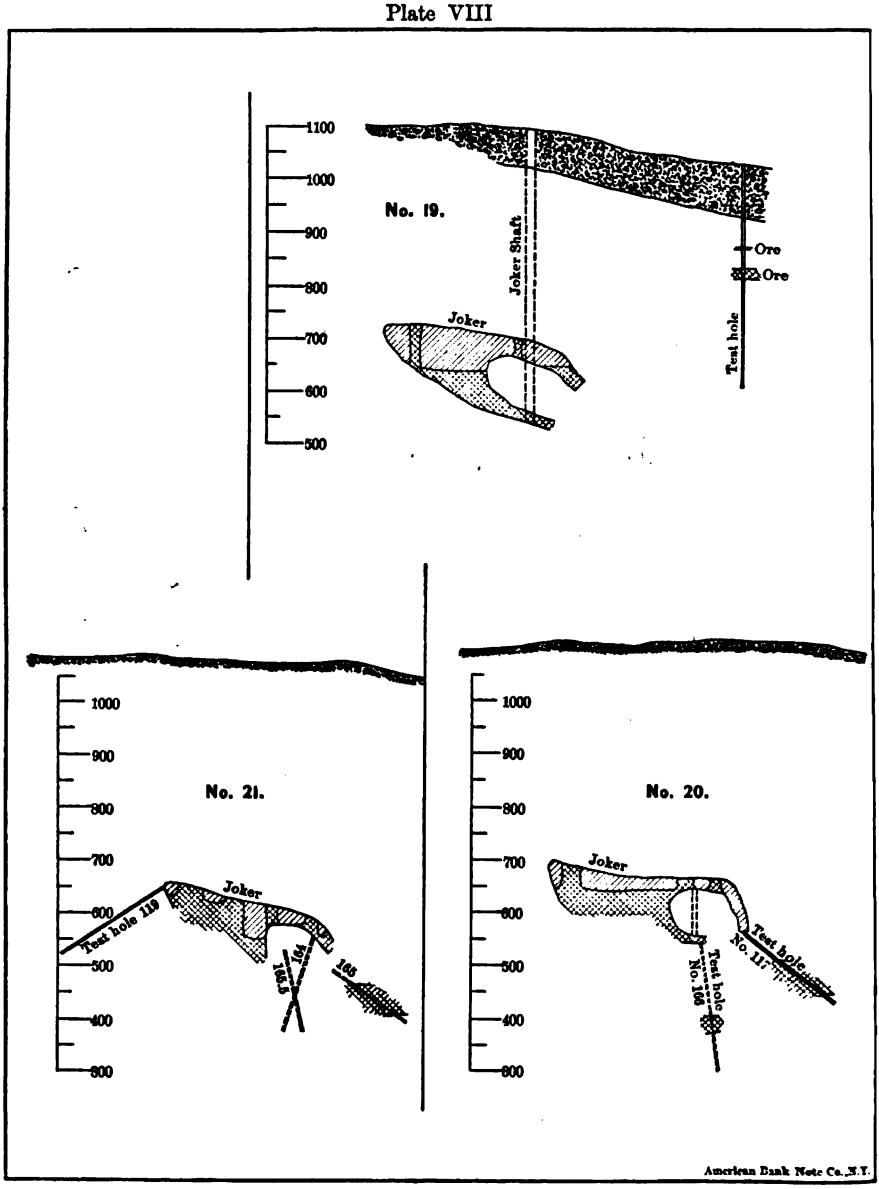
Sections 15 and 16 of the Mineville Group. See Plate II.

Plate VII



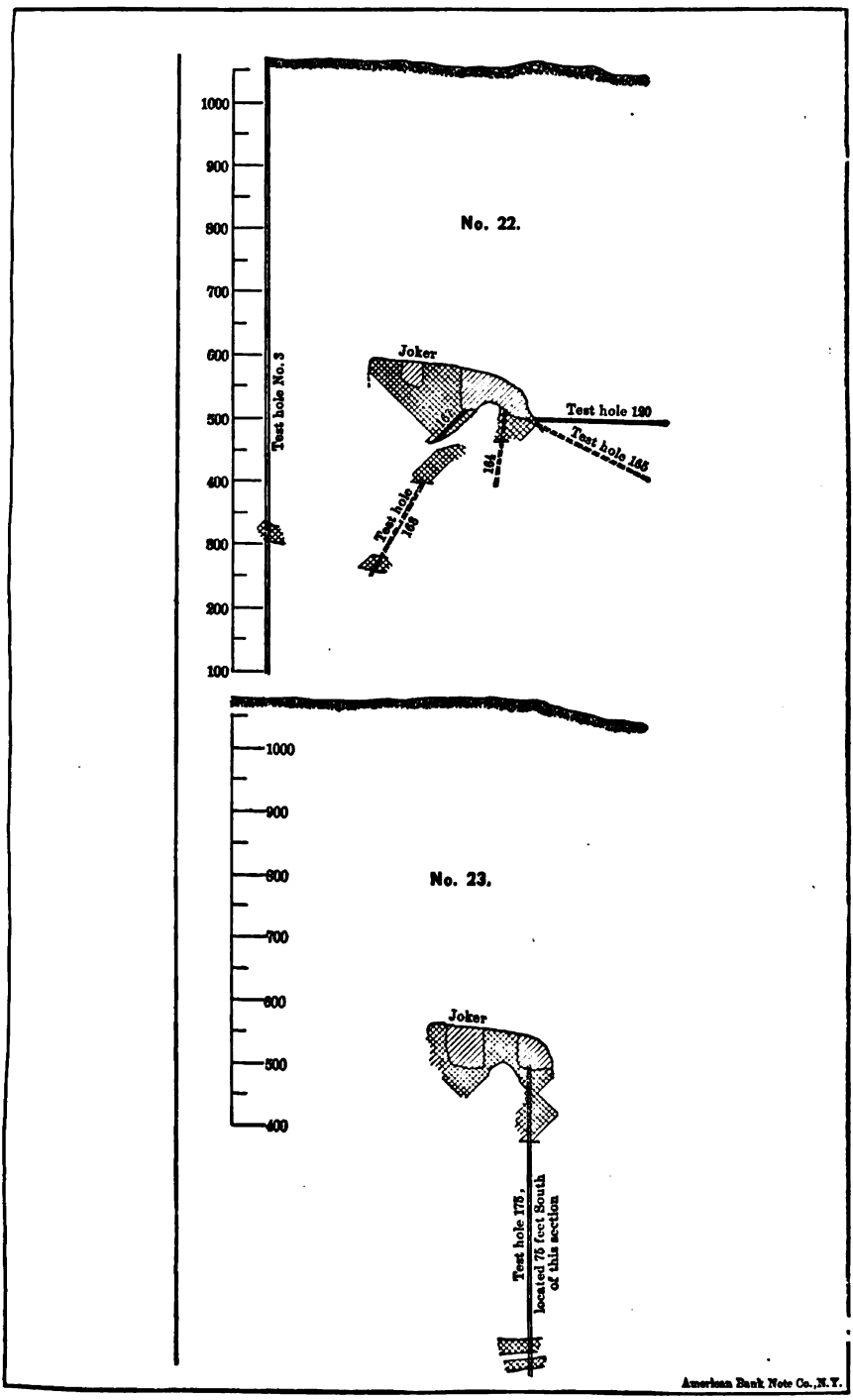
Sections 17 and 18 of the Mineville Group. See Plate II.

Plate VIII



Sections 19 and 20 of the Mineville Group. See Plate II.

Plate IX



Sections 22 and 23 of the Mineville Group. See Plate II.

ings, for, as shown in Section 7, the foot-wall of the bottom portion is at an altitude of 1000 feet; in No. 14 it is still above 900, and in No. 18, at its termination, it is 800. In its northern portion the dips are more pronounced. Its total length is over 1200 feet, its greatest thickness about 140 feet, and its greatest breadth about 300 ft. Its frequent double character and greater thickness seem to be better explained by the forking and swelling of a pod-like body than by folds and bucklings. When the sections were drawn the lower portions, from about Section 11 downward, were full of water, and the mine had long been worked out. The lower ones have, therefore, been compiled from the mine maps. The upper portions are quite easily accessible, and are in many ways the most impressive of all the local mines. The huge, dimly-lighted stopes, 100 feet and more high, impress the observer as would some vast cave.

B. T. Putnam, who visited the iron mines of New York for the *Tenth Census*, and to whose singularly careful work we owe a great debt, as regards the entire State, figures a cross-section of the Miller pit which he interprets as an S-fold or buckling. The same view is taken by Pumpelly, from Putnam's observations. (See *Tenth Census*, xv., p. 108, Pl. xxvii, Section E. Also p. 7.) The full suite of sections lead the writer, however, to the above conclusions. Putnam also gives several sections of the neighboring pits in varying directions.

Old Bed or Mine 23.—Old Bed begins on the north, at the Brinsmade shaft, as a rather thin, but somewhat steeply-dipping bed, whose inclination becomes steeper in depth. Ore still remains in the lower workings, but at this end the mines have not been in operation for years, not at all during the writer's visits, which began in 1888. Passing to the south along the strike, the Welch shaft is met that gives name to the ore-body in this portion. It is quite certainly a part of Old Bed proper, although its connection is best manifested further down. In the northern portion, the workings are among the oldest of the group and have in large part fallen shut. Sections 1, 2 and 3 are not based on as good data as are the others. In Section 6 Old Bed proper outcrops and has been mined in a great open cut. Ore still shows in the north face in several rather narrow streaks in the midst of hornblendic gneiss, and considerable ore is still remaining in the south face of the open cut, but concealed by

débris. The Welch ore-body, as outlined by the workings, passes under this part of Old Bed proper, flattens greatly in dip, and begins to exhibit the irregularities of shape that later develop in a very marked degree. Its connection with Old Bed is indicated in Section 9. As shown by the conventional signs in Sections 8 and 7, the workings have not reached the connecting portion. Old Bed proper soon develops as a great, thick pod that lies quite flat, but that gradually pitches to the south, very much like the Miller pod. It is cut off on the northwest by a well-developed fault that shows as a crushed and decomposed mass of the hanging-wall gneiss. Some bucklings or forkings of the ore into the roof are to be noted in the mine, although not apparent in the section. In Section 8, the Welch pod has turned sharply down to a vertical dip, and in Section 9 it holds the same, but has been shown to be connected with Old Bed proper. Thence, with increasing irregularity of shape, the two develop, first, the anvil-outline of No. 10; then pinch and fork into two parts as in No. 11; break apart in No. 12, and in the lowest workings are only known as rather small pods, not much opened up yet by the stopes. Old Bed, including the Welch pod, is therefore known for about 1200 feet on the long axis. It developed in the stopes, shown in No. 8, nearly 100 feet across, and where the dip becomes vertical in Section 9, or where the double fold occurs in Section 10, it had a vertical height exceeding this. The cross-sections give good ground for the inference that in Sections 8, 9, 10 and 11 a sigmoid fold has developed which has pinched out the beds into the curious shapes shown in the diagram. If we consider it as a folded and compressed single bed, it is necessary to regard the Welch bed as a prong on the Old Bed that ran off to the northeast. The peculiar warping or twist that has, however, given the Welch a westerly dip in the general direction of the Miller, while the more southern parts in the Old Bed incline southeast, following "21," is very striking. In Sections 10 and 11 there is evidence of a fault cutting off the ore on the northeast side, but its prolongation in Nos. 9 and 8 is not clear and was not noted underground. These portions of the mine had long been abandoned when the sections were made, and the dangers of an old roof made them uninviting for exploration. In Section 16, Drill Hole 170, it will be noted, has cut a double

bed of ore, about 160 feet below the bottom of the Old Bed-Bonanza pod. The upper bed is 18 feet and the lower 24. The same have been also shown, by a series of other holes, extending as far as the sections have been drawn (compare No. 23, Hole 175). It is possible, as Mr. McKee has suggested, that these may represent a continuation of the unmined portion of Old Bed, as shown in Section 8, despite the fold or forking to join the Welch extension, as shown in Section 9. One other intermediate pod, 9 feet 6 inches thick, has been cut not far from the line of Section 18, and lying a little less than 300 feet beneath the floor of the Bonanza. It is also quite probable that these beds of ore may be separate and distinct bodies lying lower in the series than the large ones.

Relations of Old Bed to Miller Pit.—There is strong evidence that the Miller is the original prolongation of Old Bed, now separated by the fault shown in Sections 6 to 12. The long, and sometimes double, chute of red-ore (phosphatic magnetite) that lies between the fault and the dike is, however, higher in phosphorus than is the usual run of the pods on either side of it, but the same ore is well recognized in "21." The apparent sudden termination of the Miller, in Section 8, against the trap dike may be due to incomplete development of the workings. Old Bed itself is thickest opposite this point. Admitting this faulted relation, a comparison of Sections 8 and 10 with 9, shows a variable amount of displacement, it being greatest in 9 in the middle and less in 8 and 10 at the extremes.

The "21"-Bonanza-Joker Ore-body.—This is much the largest of the group and is now the only one mined. It has reserves for many years to come. It is first seen in Section 6, at its northeast end, but in this portion it has been but little developed. Magnetic attraction indicates its extension for some distance further. It is a double bed, with a steep dip to the southeast, the two parts being separated by about 100 feet of rock. Of the lower bed but little is actually known along Section 6, but presumably it is the variety rich in red apatite, called red-ore, and found in its extension in Sections 7 and 8. The shaft formerly used in Section 6 is called the Tower shaft, but it has been long abandoned. In Section 8 the two beds are still separate and both have been mined. As shown on the horizontal plan, Plate II., there is an abandoned shaft, the New

Nolan, between Sections 7 and 8, and another, the Old Nolan, between Sections 8 and 9, but both have been long disused. When Section 9 is reached, the horse of rock that separated the two beds in the earlier sections has pitched down below the surface, so that their connection over its top is manifest. They are seen to be parts of one great body that has either forked around a huge horse of gneiss, or that has been folded, overthrown and compressed about it. The doubling of the ore gives a horizontal width of 300 feet of ore, and a vertical thickness in this section and in Section 10 of more than this. From Section 9 on through Section 12, the gradual pitching of this horse of rock downward about 50 feet in the hundred is very clearly shown, and furnishes one of the most interesting features of the ore-body. In Section 12 the northwesterly portion of the compressed fold begins to show a great lenticular extension, apparently pinched off from its crest. This runs through Section 14, but disappears entirely before Section 15 is reached. It is a curious mass, being shaped like a squash or a great disc, with a flat top and dome-shaped bottom. It was about 250 feet in diameter, and at its maximum was about 125 feet thick. It has all been mined out except three central pillars, and has left a huge and very impressive chamber. This ore-body is specifically known as the Tefft shaft.

The entrance of the projection of foot-wall, shown in Section 14, cuts off the Tefft-shaft body, and from this onward the northwesterly portion of the Bonanza-Joker is the edge of this lower shelf of ore, which first showed itself as a bulge in Section 12. The general form of the ore-body, viz., a compressed fold around a core of rock, is continued through Section 23. Soon after No. 23, a cross-fault comes in which has apparently dropped the roof, so as to cut off the ore across the pitch or axis of the pod. The pitch of about 50 feet in the hundred is resumed in the last sections, and it is probable that the same horse of rock first met continues to the end. In the last three sections, Nos. 21, 22 and 23, it manifests some irregularities, as shown by the drill-holes, and warps around to a vertical or northwesterly dip.

The southeastern limits of the "21"-Bonanza-Joker ore-body have, in no case, been determined. So far as the deepest workings have gone in this direction the ore is still found. While

in the earlier sections it shows a steep dip, in the later ones this seems to flatten decidedly. Quite recently the interest in this portion has been greatly increased, because three diamond drill-holes, ranged approximately parallel to the long axis of the ore-body, have shown a surprising development of ore, enough, in fact, to rival the huge cross-sections opened in the "21" pit itself. These appear in Sections 13 and 16, and indicate a downward bulge in the foot-wall sufficient to give in the former a cross-section of clear ore of over 300 feet, and in the latter of over 250. The deep hole in No. 18 showed that, at that point, the bulge had ceased, but we have no data as yet as to the relations of this ore-body to the southeast of the location of the hole. The slanting holes in Nos. 20 and 21 indicate a great prolongation on the lines of these sections, and, as further reserves, the double lower lying beds, shown in Nos. 16 and 23, are very serious factors.

At about the point where Section 24 would be located a fault cuts off the ore with an abrupt wall of "21" gneiss, and apparently the continuation of it has dropped for some unknown distance. As will be shown by reference to Plate X., and by comparing the surface contours with the end of the plotted workings, the fault comes immediately below a narrow gulch, of which it has probably been the determining factor. The general experience in the mining of these large ore-bodies in the Adirondacks has been that faults are met cutting off the ore under gulches.

Summary.—In summary, the "21"-Bonanza-Joker ore-body is essentially a double one, with a great horse of rock between. It is proved for about 1800 feet on the long axis, and is then cut off by a fault. The pitch of the crest of the ore from No. 11 (1100 feet A. T.) to No. 14 (1050 A. T.) is slight. That of the lower shelf, which first appears in No. 12 at 800 feet A.T., and is last seen in No. 23 at 575 feet A.T., is, therefore, only 225 feet in 1100. The general axial pitch of the ore-body is, thus, low in the later sections, and corresponds to the facts already recorded for the Miller and Old Bed. It is steeper in the earlier sections. The interesting structural question is whether the ore-body is a compressed fold or a forking pod, and what its relations are to the two earlier described. If the series is a single one that was once connected and approxi-

mately horizontal, it must have experienced very violent compression and doubling up upon itself, together with faulting on an extended scale, in order to bring about present relations. If, on the other hand, a series of ore-bodies, with swells and pinches, with forkings and horses, and all much pinched by dynamic metamorphism, and faulted to a moderate degree, is conceived, the case is simpler. The former view, *i.e.*, that the ore-bodies were a faulted series of a single great deposit, was taken by Prof. Hanns Hoefer during his visit in 1876, and a plan and section of the mines are given by him in accordance with this conception.* The extended series of sections now available, however, sets the facts in clearer light, and it may be stated that without them on the ground it is very difficult to keep the relations all in mind.

O'Neill Shaft and Cook Shaft.—These are located north of the Mineville group, but appear to be along the same general horizon in the gneisses. Both have been abandoned and dismantled and were full of water during the writer's visits, although the Cook shaft was working during Putnam's visit for the Tenth Census, and for some time afterward. It is called by him the Smith mine, and is stated to have an ore-shoot 140 feet wide and 25 feet thick in the middle, and pitching to the southwest at an average angle of 30°. The general trend is, therefore, parallel to the other mines. Smock states that both these shafts are connected underground.

The ores were, in each case, attacked by shafts which were several hundred feet deep. At O'Neill shaft the surface is covered with drift, but the acidic and basic gneisses are met on the dumps. The hill that rises north of O'Neill shaft is gabbro and gabbro-gneiss. Cook shaft lies on the east side of the hill, and Thompson shaft is a little north. Above it is gabbro and gabbro-gneiss, but the dump shows acidic gneisses as well as gabbro-gneiss. Considerable drift was penetrated by the shaft before bed-rock was reached. Further north than Cook shaft, and on the same hill, is the Humbug shaft, which cut the Humbug vein of titaniferous ore. In the valley still further north, and in acidic gneiss, is the abandoned Sherman mine, showing at the surface about 6 feet of workings, which dip 35° west.

* *Die Kohlen und Eisenerz Lagerstätten Nord-Amerikas*, Vienna, 1878, 175–179, Pl. IV., Figs. 14, 15.

The neighboring surface is a very complex mingling of gabbro, gabbro-gneisses and acidic gneisses.

The Barton Hill Group.—The ore of the Barton Hill group outcrops first at the south foot of the hill on the 1320 contour. At this point are located the New Bed mines, with the deepest workings in the region. The slope runs down some 2200 feet to the heading. The dip of the skip-way is about 25°. The ore-bed is a treble one at the surface, whose cross-section is illustrated by Putnam, *Tenth Census*, xv., p. 109, Section A. Next under the upper ore is a biotite gneiss that is a frequent minor associate of the Barton Hill ores, while over it is the usual Orchard gneiss. The biotite gneiss has light acidic streaks in it that consist of quartz, strained plagioclase, microcline and a little reddish biotite. Over the lower ore-bed is a mixture of quartz and magnetite, while below all is the gabbro-gneiss.

Underground these beds run off as long chutes of ore, whose axes have a general parallelism with each other and with those of the Mineville group. The workings have various names, which are from south to north: Little Pit, Big Pit, Dalton Pit, Walder's Pit and Wasson's Pit. Putnam, in the *Tenth Census*, vol. xv., p. 109, gives a section at the outcrop as visible in 1879. It shows three beds of ore; the upper is Walder's, the middle Wasson's and Dalton's, and the lower Big Pit. Big Pit has been kept open of recent years, but two of the others are full of water.

Passing up the hill, the Old North Pit is met at the 1460 contour, just beyond which is the Arch Pit. The dip increases very much, and at the latter is 45°. The hill is quite steep on the south side at this point, as the surface rises to the shoulder shown in the contours, Plate X. The next pit is the Lover's Hole. The ore lies quite flat at the surface, but increases to a 15° dip, and soon thereafter to a much steeper inclination, where the bed, previously single, forks into two, the lower of which soon widens to a lens about 20 feet thick by 60 feet across, that yielded phenomenally rich ore and remarkably large and perfect octahedral crystals of magnetite.* Forty

* John Birkinbine, "Crystalline Magnetite in the Port Henry, N. Y., Mines," *Trans.*, xviii., 747. A good cross-section of the Lover's Hole Pit is given in this paper. See also J. F. Kemp, "Gestreifte Magnetitkrystalle aus Mineville, Lake Champlain Gebiet, Staat New York," *Zeitschrift für Krystallographie*, xix., 183.

thousand tons were obtained that averaged 68.60 Fe and 0.033 P, being the richest large lot of ore ever mined in the east. Above the Lover's Hole, after an interval, are the South Pit, the North Pit and the Orchard Pit, with dips of from 15° to 19°, and all single beds at the outcrop. Beyond the Orchard is a stretch of 2000 feet, without openings, before the Fisher Hill mines are met.

At Fisher Hill and Burt Lot the relations are somewhat more complex, and in some cross-sections three different beds are apparent. The ore has a general dip of 25°. At Fisher Hill the outcropping beds are about 100 feet apart as a maximum horizontally on the surface, which would be 40–50 feet vertically, or a little less, proportionately, perpendicular to the foliation. At Burt Lot the maximum interval on the surface is more, being 250–300 feet, and the vertical separation for the former value would be about 116 feet, and, between beds perpendicular to the foliation, 10 feet less. There seems, however, no pronounced regularity of horizons, but rather the presence of pods of ore along approximately the same horizon and near the contacts of the gabbro-gneiss and the Orchard gneiss. At Fisher Hill both walls on the surface are the former, and at Burt Lot they are the latter. How much further the belt runs to the north is not known, but strong attraction indicates its presence for some distance.

The Chemical Composition of the Ores.

The ores of the Mineville group, as well as those of the O'Neill and Cook shafts, are all non-Bessemer, and rule rather high in phosphorus, in some instances unusually high. Putnam's samples, which are fair averages, gave the following results:

	1.		3.	4.	5.
Fe,	57.71	61.39	60.54	61.53	62.64
P,	1.266	1.282	0.830	1.302	0.908
TiO_2,	Present.	Present.	Present.	Present.	Present.
	6.	7.	8.	9.	10.
Fe,	61.92	62.10	67.38	61.39	62.66
P,	0.876	1.198	0.465	1.496	1.118
TiO_2,	Present.	Present.	Present.	Present.	Present.
		11.	12.	13.	14.
Fe,		61.46	63.01	62.66	62.80
P,		1.129	0.967	0.946	0.870
TiO_2,		Present.	Present.	Present.	Present.

No. 3 is from the Miller Pit; Nos. 1, 4 and 5 are from Old Bed and its extensions, being respectively from the Brinsmade, the Potts shaft and Old Bed proper. Nos. 2, 6, 7 and 8 are from mine 21 and its extensions, respectively, the Nolan shaft, the Tefft shaft, "21" pit weathered ore, and compact ore from same. Nos. 9 and 10 are samples from the shipping piles of the Port Henry Iron Ore Co., being respectively from 30,000 tons of lump and 40,000 tons of fine. Nos. 11 and 12 are from similar piles of Witherbee, Sherman & Co.'s output, respectively 23,000 tons lump and 38,000 tons fine. Nos. 13 and 14 represent the lump and fine ores of Cook shaft. The ores are all rich, and are exceptionally high in phosphorus, as iron-ores go. It is unfortunate that the TiO_2 was not determined, but it is metallurgically negligible, although scientifically interesting. Prof. Hoefer, on p. 178 of the report cited later in the bibliography, gives a complete analysis of Old Bed ore, as furnished by Mr. Witherbee:

Fe,	64.25
O with Fe,	24.49
H_2O and organic,	0.31
Insoluble, SiO_2, etc.,	3.24
S,	0.03
P_2O_5,	2.93
Al_2O_3,	0.80
CaO,	3.55
MgO,	0.13
MnO,	0.09
Undetermined,	0.18
	100.00

The interesting point of the titanic acid is not quantitatively solved by the analysis, as the TiO_2 was doubtless weighed, if present, with one or more of the other ingredients. The lime is chiefly from the apatite.

The Barton Hill group supply almost entirely Bessemer ores, which, in instances, have proved extremely rich and pure. Putnam's samples, and certain others as stated below, ran as follows:

	15.	16.	17.	18.	19.	20.
Fe,	43.74	39.15	49.86	46.91	68.60	62.99
P,	0.002	0.019	0.048	0.030	0.033	0.024
TiO_2,	Absent.	Present.	Present.	Present.		

	21.	22.	23.	24.	25.	26.
Fe,	63.48	63.34	63.40	66.81	65.65	65.12
P,	0.047	0.02	0.045	0.032	0.027	0.0289

No. 15 is from 5000 tons New Bed; No. 16 from 3 cars of the so-called South Barton Hill mines, *i.e.*, Old North Pit and Arch Pit; No. 17 is from 500 tons from the mines farther up the hill; No. 18 is from 8500 tons Fisher Hill ore. No. 19 is the average of 30 samples, representing 40,000 tons of ore from the rich chute in the Lover's Pit; it is from Mr. Birkinbine's paper, cited in the bibliography. From the same source are taken the remaining analyses. No. 20 is from the North Pit, average of 26 samples; No. 21, from South Pit, average of 28 samples; No. 22, from Orchard Pit, average of 22 samples; No. 23, from Roe shaft (near New Bed), average of 4 samples; No. 24, from Big Pit (part of New Bed), average of 21 samples; No. 25, from Little Pit (part of New Bed), average of 15 samples; No. 26 is the average of Nos. 19 to 25, inclusive.

It is evident from these that, in later years, the ores have been very rich and pure.

Three complete analyses are available of these ores: (*a*) is of New Bed, and (*b*) a general sample of the Bessemer ores, both analyses being taken from Hoefer; (*c*) is of the crystalline magnetite of the Lover's Pit, and is taken from Birkinbine's paper, as already cited:

	(*a*)	(*b*)		(*c*)	
Fe,	68.24	50.38		71.58	
O with Fe,	24.49	19.20		27.27	
H_2O and organic,	0.38	0.54			
Insoluble siliceous,	4.32	29.11			
S,	None.	None.		None.	
P_2O_5,	0.038	0.05		0.009	
Al_2O_3,	0.028	0.32	0.240		0.240
CaO,	0.14	0.17	0.280		0.310
MgO,			0.350		0.330
MnO,					
Undet.,	0.592	0.23			
SiO_2,			0.110		0.125
TiO_2			0.147		0.150
	98.228	100.00	99.936		100.014

These analyses give us a very interesting insight into the composition of the ore, although we cannot but regret that the insoluble matters in (*a*) and (*b*) were not broken up. In (*c*) the FeO was found to be, in duplicate, 23.727 and 23.753, being very near the theoretical value. The titanic acid is also of interest, as it shows the quantitative amount of this substance in

very pure material. The slight impurity present was doubtless a little chlorite and serpentine in the cleavage cracks of the mineral.

The Burt Lot ores run comparatively low in iron, although of Bessemer grade. They are chiefly adapted to concentration, for which the large mill at Mineville was built. Recent conditions of the iron business have precluded its use.

The Geological Relations of the Ores.

The map, Plate X., is intended to illustrate the geology of the area embracing the mines. Only outcrops are plotted, and, where no rock is indicated, the surface is buried in glacial drift. It is at once evident that the mantle of the latter is widespread, and that comparatively few ledges project except on the hill-tops and at the mines. Five different rocks are distinguished in the signs, of which the fifth is a massive, unmetamorphosed form of one of the other four. For clearness in description, these will be called the "21" gneiss, the Orchard gneiss, the Barton gneiss, the gabbro-gneiss and gabbro. All except the most massive variety of the gabbro display gneissoid foliation, but in the "21" gneiss and the Orchard gneiss it may be faint.

The "21" Gneiss.—This appears in its typical exposure in the hanging-wall of the great pit at the "21" mine. It can be obtained on the south side of the pit, as well as in the workings. It is a moderately coarsely crystalline rock of pronounced granitic character. To the eye it exhibits almost no dark silicates, and appears to be little else than quartz and feldspar. Near the ore, just as has been the experience at Hammondville, little red crystals increase noticeably in amount and suggest garnets, but the thin sections indicate that they are titanite.

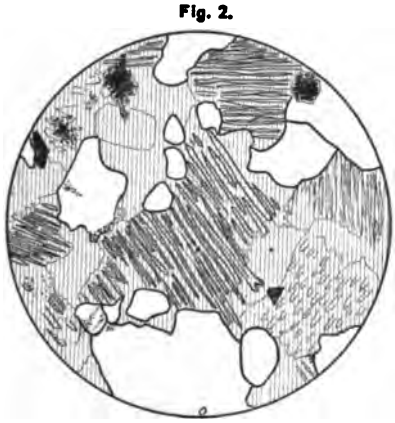
Under the microscope, the rock is seen to consist of microperthitic feldspar and of quartz, together with smaller and scattered grains of plagioclase, magnetite, titanite and zircon. A typical case is shown in Fig. 2,* which will give a fair idea

* The five microscopic drawings were all made from photomicrographs, that is, from photographs taken with a camera whose lense was an ordinary petrographical microscope. Blue-prints were then made from the negatives and inked in on the lines of the photograph with water-proof ink. The blue-prints were then placed in a weak solution of sodium carbonate, which bleached them, except for a faint

of the relations. It may be well to explain that microperthite is a name applied to a variety of feldspar, particularly orthoclase, that has as inclusions flattened spindles of some other feldspar, especially albite, in parallel alignment, so that the latter may even constitute the larger part of the aggregate.

The microperthite occurs in irregular grains and at times contains rounded inclusions of quartz, indicating a succession of the minerals in order of formation the reverse of that in the usual igneous rock, the quartz, if it is a real inclusion, having formed before the feldspar. These relations are common in metamorphic rocks and lead one to infer that the gneiss has

Fig. 2.



"21" gneiss; ordinary light; actual field, 2.5 mm. = 0.1 in. The clear mineral is quartz; the shaded one feldspar. The shading is purely conventional. The spindles of albite in the micro-perthite are self-evident. The black is magnetite.

gone through some process of recrystallization. During this, the quartz and feldspar may have crystallized at the same time, or the quartz may have resulted from some secondary change in the original feldspathic compound. Microperthite itself is more common in metamorphic rocks than in unchanged igneous ones, although it appears to be not unknown in the latter.* It

brown tinge, and afterwards in very dilute hydrochloric acid, which turned the brown pale blue, leaving the black ink on a white or pale blue (actinically white) ground. In this way the accuracy of a photograph is preserved, together with the distinctness of a pen drawing. Photomicrographs of rock sections are otherwise very unsatisfactory because of their flatness and failure to bring out desired details. The drawings were kindly made by the writer's friend, Dr. W. D. Matthew, of the American Museum of Natural History.

* Osann has recorded microperthitic orthoclase in a phonolite from Western Texas, which with some other peculiarities led him to give it the special name paisanite. Tschermak's *Min. und Petrog. Mittheilungen*, xv., 435.

seems to be due in the present instance to the penetration of solutions bearing soda into the original feldspathic component, in this case supposed to have been prevailing orthoclase, and to the formation of albite by their reactions with it. All the minerals exhibit severe dynamic strains as indicated by the undulatory extinction and the soda solutions probably followed solution planes, made accessible by the strains. The albite spindles are not parallel to either of the cleavages, but cut them at varying angles, according to the location of the section. It is fair to state that the albite may be explained as due to the breaking up of a soda-bearing orthoclase into albite and orthoclase, but the former explanation is regarded as the more probable. The microperthite varies in degree of development, and begins in the central portion of the crystals. It is often surrounded by a border of clear feldspar. It is certainly an indication of considerable metamorphic change. It figures prominently in the Barton gneiss, and its faint beginning may be detected in the massive gabbro.

The plagioclase is normal, multiple-twinned plagioclase, whose extinctions are those of oligoclase. The other minerals are unimportant.

The Orchard Gneiss.—This is found in typical development as the hanging-wall of the Barton Hill ore-beds and appears all along their outcrops as a narrow band 30 to 50 feet across. It is also met in diamond-drill-holes almost invariably over ore-bodies, and forms a considerable outcrop on the slopes of Barton Hill above the Miller Pit.

In the hand specimen it looks very like the "21" gneiss. It is light-colored, granitoid in texture and shows practically no dark silicates, quartz and feldspar being the only minerals recognizable. In thin section however, it is found to contain quartz and plagioclase alone, no microperthite being present. The plagioclase is shown by its optical properties to be oligoclase. Small magnetites and zircons are also present. In one slide (511) from a band of this gneiss, lying between two of the ore-beds at New Bed, considerable microcline is met.

The Barton Gneiss.—This forms the summit of Barton Hill. It is a decidedly darker gneiss than either of the others just described, and contains abundant dark silicates. It begins about 50 feet, more or less, above the Barton Hill ore-bodies and

extends to the summit. It is also met in drill cores over toward Mount Tom, as shown by the cross-sections CC and DDD. In mineralogy it is intermediate between the " 21 " gneiss and

Fig. 3.

Orchard gneiss; ordinary light; actual field, 2.5 mm. = 0.1 in. The colorless minerals are quartz and plagioclase, each of which is brought out in polarized light (Fig. 3a). The two crystals with heavy edges are zircons; the black mineral is magnetite.

the gabbro. Under the microscope it exhibits microperthite as the most abundant mineral, with which are small and less

Fig. 3a.

Orchard gneiss: crossed Nicols; actual field, 2.5 mm. = 0.1 in. The striated mineral is plagioclase; the clear one quartz. The concentric lines parallel to the edges are due to strains in the crystals brought out by polarized light.

frequent quartzes, oligoclases and orthoclases. The dark silicates are brown hornblende, emerald-green augite, and rarely hypersthene. All are in irregular pieces or, to use Pirsson's

useful term, in anhedra.* Apatite is common and the anhedra are moderately large for this mineral. A good average specimen of it is shown in Fig. 4. Quartz is less abundant than in either of the previous gneisses.

The Gabbro and Gabbro-Gneiss.—These two rocks, which differ considerably in mineralogy and texture, so much so that the massive variety can be readily distinguished from the gneissoid in the hand specimen, are nevertheless described together, because they are essentially one geological unit and pass one into the other by insensible gradations. This is a familiar phenomenon in the Adirondacks, and, as is well known, variability is a characteristic feature of gabbros.

Fig. 4.

Barton gneiss; ordinary light; actual field, 2.5 mm. = 0.1 in. The clear minerals are quartz and microperthite. The microperthite is the one surcharged with little spindles. The minerals faintly shaded are orthoclase and oligoclase. The minerals with the heavy borders are hornblende when shaded and augite when not. Magnetite is dead black; and apatite is faintly dotted.

The massive gabbro is illustrated in Fig. 5. It is a moderately coarse, granitoid aggregate of plagioclase (labradorite, and, from the optical properties, some even more basic members of the plagioclase series), pale green augite, hypersthene, brown hornblende, rare olivine, garnets, magnetite and apatite. The plagioclase is in rather broad anhedra, often untwinned, but when in this condition exhibiting a biaxial interference figure in characteristic relations to the cleavage cracks.

* By anhedron (plural anhedra) is meant a component mineral in a rock which is irregularly bounded and without the plane faces implied by the word crystal. It was suggested by L. V. Pirsson, *Bulletin Geol. Soc. America*, vol vii., 492, 1895.

The dark silicates are more often in granular aggregates than in larger anhedra. A grain of magnetite may form a nucleus, around which will be a rim of brown hornblende, and then the feldspar; or a group of hornblendes, augites and hypersthenes may occur together. The olivine is in rare and serpentinized anhedra. The garnets form, as a rule, slender, finger-like aggregates in the feldspar. They may ramify irregularly or may follow certain of the twin lamellæ of the feldspar, presenting very curious and interesting phenomena. Some of the feldspars show microperthitic development, with the small albite spindles sparsely developed in the midst of an untwinned an-

Fig. 5.

Massive gabbro; ordinary light; actual field, 2.5 mm. = 0.1 in. The light mineral with parallel lines is labradorite; the one with heavy parallel or diagonal shading is hornblende; the one with rectangular cleavage-cracks is augite; the irregular grains with heavy borders are garnet, and the black is magnetite.

hedron, which extinguishes just about parallel to its cleavage cracks, and may be orthoclase, although the extinction is undulatory. The spindles themselves make an angle of about 70° with the cleavage. The twinned plagioclase also shows in many cases the dusty inclusions in the center of an anhedron, that fade out toward the borders. These are probably pyroxenic dust and are well-developed in many Adirondack gabbros, and have been previously noted. They are quite common in other regions as well in Canada as in Europe. The development of the microperthitic feldspar is highly significant, occurring, as it does, in a plutonic gabbro.

There are several good exposures of the massive gabbro, but

perhaps the best is under the second wire-tramway support just below the Lover's Pit. The gneissoid variety is the commonest rock along the lower slopes of Barton Hill, and embraces the massive exposures in its larger area. It is a dark, coarsely laminated rock that exhibits under the microscope coarse, brown hornblende and more or less kaolinized plagioclase, with practically little else. Intermediate varieties with considerable augite and with a little hypersthene and garnets have also been met, but with the passage of the gabbro into hornblende gneiss, the pyroxenes and the garnet disappear. The change is thought by the writer to be due to dynamic metamorphism, and has been previously noted.* A typical thin section is illustrated by Fig. 6.

Fig. 6.

Gabbro-gneiss; ordinary light; actual field, 2.5 mm. = 0.1 in. The clear mineral is plagioclase; the one with diagonal cleavage is hornblende; the black is magnetite.

Pegmatites are not lacking in the gabbro-gneiss in streaks as a rule small, and one vein of orthoclase was met just north of the Lover's Pit on the slope of the hill.

Minor Rocks.—The excavations and dumps afford other varieties of rocks not included in the above. On almost all the dumps along the Barton Hill group a richly micaceous gneiss is afforded in which biotite is a prominent constituent. Under the microscope it shows in addition emerald green augite, green hornblende, plagioclase, magnetite, apatite and unusually abundant titanite. It is regarded as a phase of the foot-wall gabbro-

* J. F. Kemp, "Gabbros on the Western Shore of Lake Champlain," *Bull. Geol. Soc. of America*, v., 221, 1894.

gneiss near the ore. Near the ore are also met coarse aggregates of hornblende, plagioclase, magnetite and quartz in the nature of pegmatites. With these are found some of the most interesting minerals of the locality, as will be later noted under the paragraph relating to the mineralogy of the mines. These hornblendic aggregates make it difficult, if not impossible, to get the true foot-wall in the mines. At the Lover's Pit is also found a dense green rock, which contains pockets of pink and green fluorite. It is an aggregate of grains of quartz, a half-millimeter (0.02 of an inch) in diameter and surcharged with a felt of little needles. The latter are double refracting and have extinction angles ranging up to 20°. They are probably actinolite, but they are so excessively minute that more reliable data were not obtained. The quartz was determined on the basis of the positive uniaxial character of the sections.

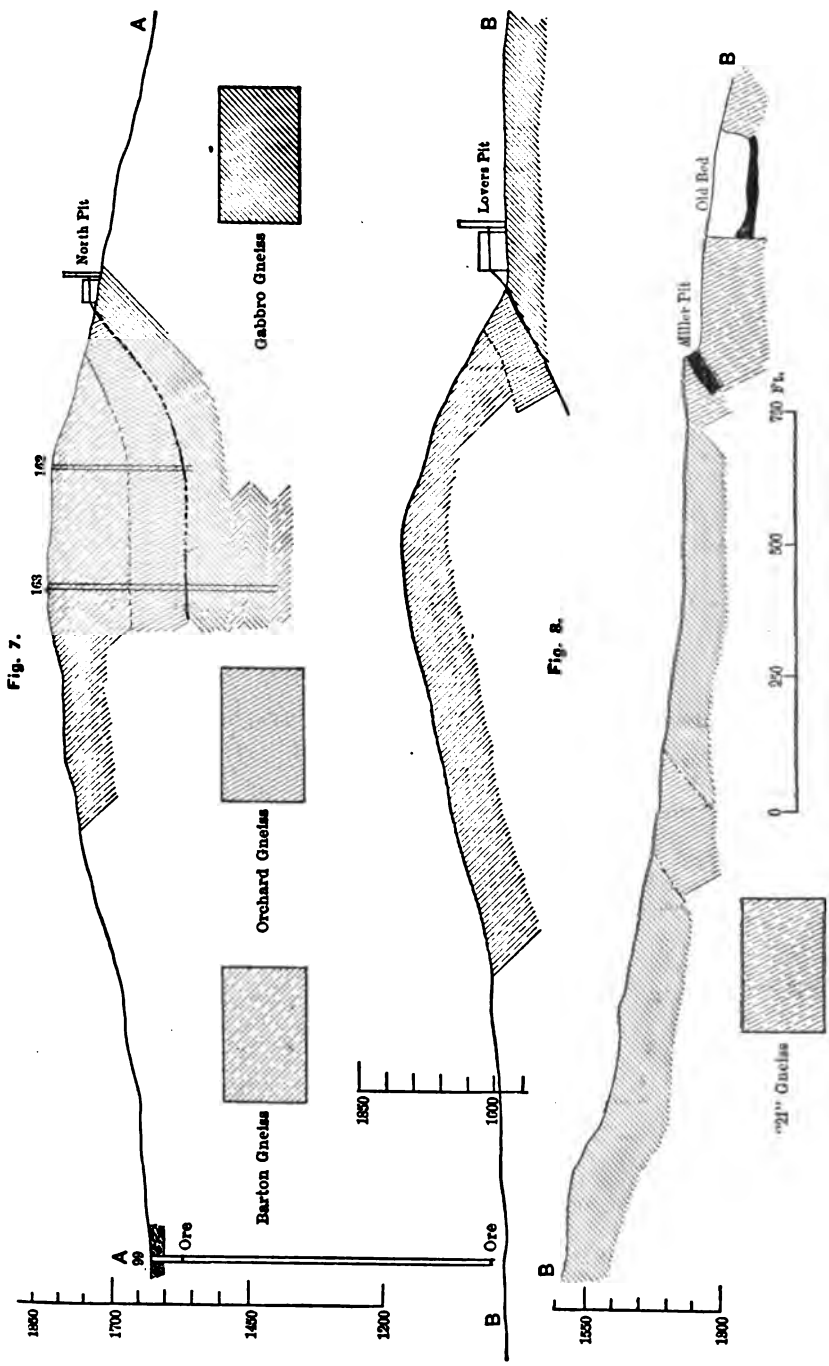
In Mine "21" coarse pegmatite consisting of orthoclase, albite, quartz, hornblende, biotite and magnetite form a large part of the horse of rock over which the two portions of the ore seem to be folded. Notes on the albite, whose identity was a surprise, will be found under the heading Mineralogy. The portion of it that was hoisted for the dumps last summer (1896) was richly provided with zircons and other interesting minerals, as later noted. From the sides of the great pit, veins or dikes of pegmatite can be seen cutting into the ore, appearing to be of later formation than it. The trap-dike earlier referred to in the Miller Pit is diabase.

Mineralogical Relations of the Gneisses.—It is evident from the preceding descriptions that in their minerals the several gneisses are more or less related. The pure plagioclase-quartz aggregate of the Orchard gneiss passes into the "21" gneiss by the addition of microperthite, which is regarded as a secondary metamorphic product, some plagioclase being always present in the "21" gneiss. The entrance of bisilicates causes the difference between the "21" gneiss and the Barton gneiss, and the decrease in quartz accompanies this increased basicity. It is possible that all these are varying phases of the same original rock mass, and the general characters suggest its igneous affinities. From these three the gabbro and gabbro-gneiss are easily set off as later and quite different members of the series.

The Stratigraphical Relations of the Gneisses.—By reference to

the map, Plate X., it will be seen that the "21" gneiss is exposed over the large pits, and especially over the "21" pit itself, and on the south side it forms a wall about 100 feet in thickness. The outcrop of the same rock over the Miller Pit quickly gives place to the gabbro-gneiss as one proceeds northwest. This extends for 250 to 300 feet, and then in turn yields to a tongue of the Orchard gneiss, about 100 feet across, but this latter pinches out to the southwest, being entirely surrounded at this end by the gabbro-gneiss. The northeasterly extent is unfortunately all concealed by glacial drift. Beyond the Orchard gneiss a steep shoulder of Barton Hill soon rises, entirely formed of the gabbro-gneiss, but to the southwest it shades into the massive type. The type specimens of the massive gabbro were collected in this area. Resting on the gabbro-gneiss is the ore, sometimes in one, sometimes in two or even three separate beds, and of irregular shape, but exhibiting in the pods or chutes a marked axial alignment to the southwest. Over the ore lies a rather thin but very persistent stratum or sheet of the Orchard gneiss, roughly fifty feet thick, and then the Barton gneiss forms the top and western side of the hill, and is met in Belfry Hill to the west, where, however, it is often rich in large garnets. These relations appear at a glance on the maps, Plates X. and XI., and on the two sections AA and BB, Figs. 7 and 8. AA is of especial interest, because it is based on both surface observations at a place of good outcrops, and on two drill-holes, Nos. 162 and 163, which lie on its line. Samples from 163 have been microscopically determined, and have furnished one of the most important checks on the stratigraphy.

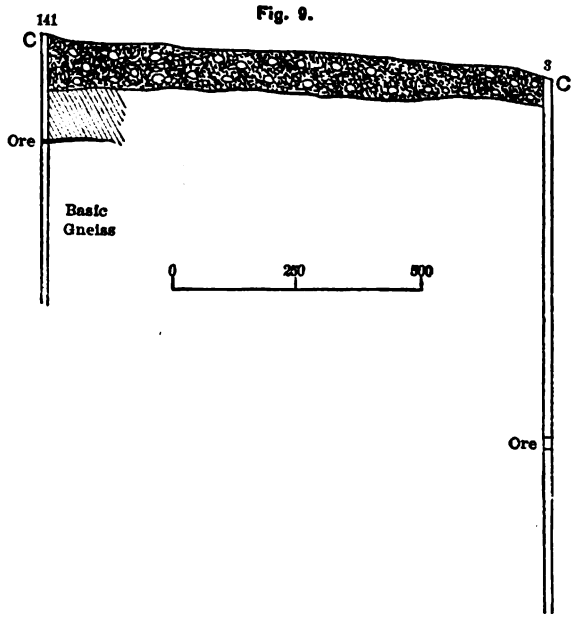
The sections exposed by the two holes are important, because they show that the contrasted sheets of gneiss are quite regular and persistent for considerable distances. In depth, moreover, in No. 163, at 75 feet below the ore the gabbro-gneiss gave place to a narrow sheet of the Barton gneiss, 25 feet thick, which was followed by the gabbro-gneiss for 62 feet, and then after penetrating the Barton gneiss for 5 feet the hole stopped. On the surface, as the small outliers show, the Barton gneiss extends both northeast and southwest. The same is true of the gabbro-gneiss.



Cross-section of Barton Hill, on Lines AA and BB, Plate X.

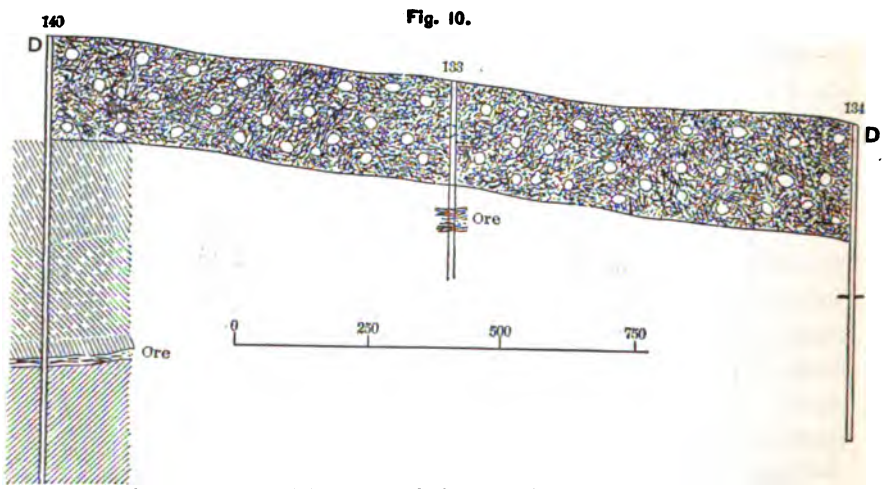
In sections CC and DDD, Figs. 9 and 10, the relations are again shown stratigraphically, the determinations in hole No.

Fig. 9.



Cross-section on CC, Plate X., between drill-holes 141 and 3.

Fig. 10.



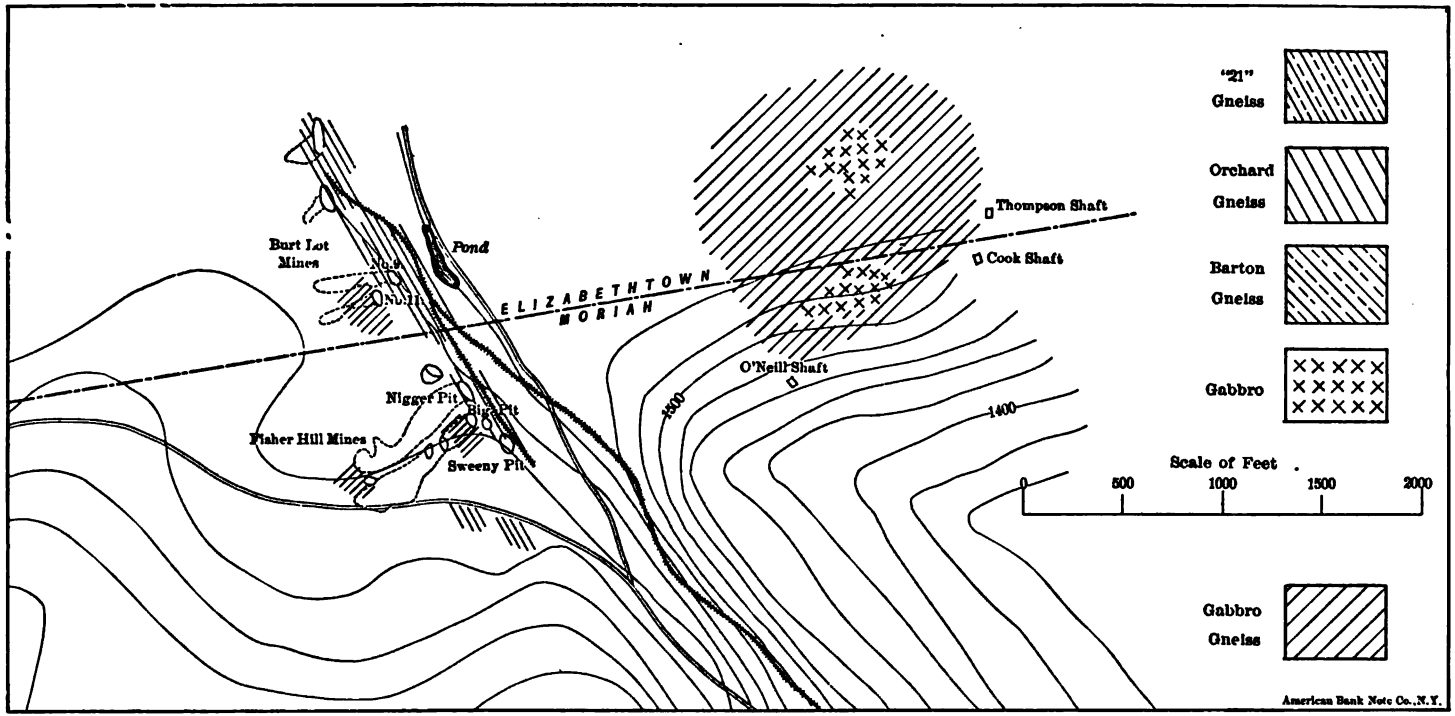
Cross-section on DDD, plate X., between drill-holes 140 and 134.

140 having been microscopically made. The others are based on names given by Mr. McKee on the basis of my determinations of the cores of 140.

Under the "21"-Bonanza-Joker ore-body, the gabbro-gneiss has been cut by drill cores, and that of hole 44 B, at 310 feet, has been examined microscopically. It is the usual aggregate of hornblende and plagioclase. So far as the drill records show, it extends without break to a 9-foot bed of ore, which was met in this hole somewhat more than 100 feet below the "21" ore-body, shown in Section No. 13, Plate V. The foot-wall of the 9-foot bed is the normal "21" microperthitic gneiss.

The geological relations at Fisher Hill and Burt Lot are more complex, although, broadly speaking, they are much the same. As shown on the map, Plate XI., the Orchard gneiss appears in the eastern pits of Fisher Hill and forms both walls. Under No. 9 a very considerable cross-section is afforded, there being quite 40 feet of it exposed below the ore, and containing small streaks of magnetite. It also appears over the ore. When, however, the next pit to the west, No. 11, is examined, a much darker gneiss is found in both walls, and one that has closer relations with the gabbro-gneiss than with any other. It is therefore drawn with the sign of the gabbro-gneiss. It consists of plagioclase, brown hornblende, emerald green augite, brown biotite, apatite and magnetite. The feldspar shows no microperthite. The rock therefore differs in this last respect from the Barton gneiss, although like it in others. It is in the stratigraphical position of the Barton gneiss, and there may be some question whether it represents it or the gabbro-gneiss. Along the highway the Orchard gneiss runs to the west of the general line of the Barton gneiss, and appears also to the northward beyond the actual pits. East of the northward extension of the highway that passes next the little pond, gabbro is extensively developed, and, as drawn on the map, it forms a large hill to the eastward. Still further to the northeast, near the little hamlet of Fletcherville, the relations of the gabbros and acidic gneisses are again very complex, and are not taken up here. On its east and south sides this hill has been productive of much ore, from the Cook shaft and the O'Neill shaft. Both are quite deep, are now abandoned and are therefore inaccessible. The dumps show, however, both basic and acidic gneisses, as is the usual rule. The ores in character and geological position correspond to the Mineville group rather than the Barton group.

Plate XI



Geological Map of Fisher Hill, Burt Lot, and the Hill to the East of them. A continuation of Plate X.

The greatest geologic interest attaches to the interpretation of the nature of these constituent gneisses. The following brief summary is an endeavor to set forth the certainties and the most reasonable inferences:

1. The foliation or gneissoid development is most prominently shown in those rocks that are rich in dark silicates. These are the gabbro-gneiss and the Barton gneiss. It almost fails in the others. In the gabbro-gneiss there is little doubt that it has been developed in an otherwise massive plutonic rock by dynamical processes, such as shearing and viscous flow under pressure. The foliation of the other gneisses has probably originated in the same way, but that they were originally plutonic or other igneous rocks is less clear, although it is the conviction of the writer that they were. This question is an obscure one, on which, however, chemical analyses might throw some light.

2. The foliation, although more or less irregular, yet follows very closely, so far as one can observe, the contacts of the different kinds of rock. On the slopes of Barton Hill it is undulatory in places, but it corresponds closely, and apparently exactly, to the strike and dip of the ore.

3. The ores lie at the contacts of the gabbro-gneiss and a much more acidic type, which on Barton Hill and in some drill-cores, to the southwest of the Mineville group, is the Orchard type of gneiss, but which in "21" and the neighboring mines is the "21" type. As a rule the gabbro-gneiss forms the foot-wall, the acidic gneisses the hanging, but the reverse is true in hole 44 B, Section 13, where, as stated in the fifth paragraph above, a 9-foot bed is intersected. The location at the contact is also less clearly shown at Fisher Hill and Burt Lot, for at the mouths of the pits the ore is in a single kind of wall-rock. At the Big Pit this is gabbro-gneiss, as determined in specimens from the hanging, but the "21" gneiss was found on the dump. At the Burt Lot mines the Orchard gneiss appears along the railway, below No. 9 pit and also in the hanging, whereas the hanging of No. 11 pit, 50 to 75 yards west, is the questionable gabbro-gneiss, and the ore appears in it. Although this reverses the usual succession, and although the ore is sometimes all in one kind of rock, yet the contact is near. It is freely admitted that difference of opinion might arise as to the deter-

mination of the darker basic gneisses, whether they are to be considered of the gabbro-gneiss type, or of the Barton type, as was set forth earlier in speaking of the stratigraphical relations, but the best interpretation seems to the writer the one just given.

4. The relations of the rocks lead to the conclusion that the gabbro was intruded as one or more sheets, which pierced the gneisses parallel with the present direction of foliation, and which probably caught up large inclusions, such as the mass of the Orchard gneiss above the Miller Pit. The intrusion under the Mineville group may be a different one from that under the Barton Hill group, and the contrasted ores give some ground for the inference, but lithologically both are the same kind of rock. Subsequent to the intrusion and to the ore-deposition came the dynamic metamorphism which developed the gneissoid foliation. Presumably the pressure acted in a northwesterly direction and caused the elongation of the ore-bodies to assume its northeasterly trend. The Barton Hill group remain in a monocline, but the Mineville group are in a faulted anticline. The ore-deposition, the metamorphism and the folding are all of pre-Cambrian date, but some of the faulting probably is later. The rocks are apparently of deep-seated and plutonic nature, as are the phenomena of ore-deposition.

The Origin of the Ores.

This is an obscure question, and to a certain extent, so long as doubt hangs over the nature of the acidic gneisses, whether sedimentary or igneous rocks in their unmetamorphosed condition, just so long will an element of doubt enter into the origin of the ore. Indeed, the method of formation of our magnetite ores, in the Archæan gneisses, has always been esteemed a very hard problem, and, although it seemed simpler in earlier years, when the gneisses were confidently believed to be metamorphosed sediments, whose bedding was preserved in the foliation, and when the magnetites were supposed to have been limonite or clay-ironstone or black-band beds, such as we see in sediments to-day,* yet the advance of investigation in

* See, for instance, J. S. Newberry, "The Genesis of Our Iron-Ores," *School of Mines Quarterly*, November, 1880, vol. ii., p. 6.

metamorphic districts has so often shown the affinities of gneisses with dynamically metamorphosed granites and other plutonic rocks, that all of our later observers have adopted a very conservative attitude regarding the ores. Many have regarded them as segregated veins whose iron oxide has collected by a leaching of the neighboring wall-rock and a concentration of it into lenticular masses parallel to the foliation (or supposed bedding) of the gneisses. It is, however, difficult to conceive of the ores coming through hard crystalline rocks in this way (unless the deposits were formed before metamorphism) and of making a place for themselves by either forcing the wall-rock to bulge laterally or by replacing it. If deposition before metamorphism is assumed, we are then brought back to the uncertain ground of what the rocks were in those early times. The lenticular shape is best accounted for by pressure and consequent stretching of a bed or mass already formed, and for magnetites not associated with rocks demonstrably igneous, the above conception of a segregated vein may properly seem reasonable.*

Others have thought them replacements of limestones, presumably stretched out into lenticular beds by dynamic metamorphism; but these magnetite mines are notably lacking in minerals involving much lime; and calcite, while occasionally met in small cross-fissures, is, as a rule, a rarity.† Its original existence is a matter of assumption. Again, conceiving of the gneisses as originally fragmental sediments, the magnetites and their associated apatite have been regarded as concentrated placers or shore-deposits of relatively heavy minerals, such as we sometimes see along large rivers or on sea-beaches.‡ This view has claims to serious attention; but, again, it is built on the uncertain foundation of the early history of the gneisses. It and indeed all the views involving this original sedimentary condition of the wall-rock, have received some corroboration from J. E. Wolff's very important and significant investigations at Hibernia, N. J.,§ where a study of the structural geology

* Compare R. S. Tarr, *Economic Geology of the United States*, p. 130.

† See in this connection J. P. Kimball, "Genesis of Iron-Ores by Isomorphous and Pseudomorphous Replacement of Limestone, *Amer. Jour. Sci.*, Sept., 1891, 231; *Amer. Geologist*, Dec., 1891, 352. Archean magnetites are not specifically treated.

‡ This conception is urged by B. J. Harrington, *Can. Geol. Survey*, 1873-4, p. 193, and by A. A. Julien, *Proc. Philadelphia Acad. Sci.*, 1882, p. 335.

§ "Geological Structure in the Vicinity of Hibernia, N. J., and Its Relation to the Ore Deposits," *Annual Report of N. J. State Geologist* for 1893, p. 359.

gave greater reason to view the gneisses as a sedimentary series having a top and bottom than anything else. Prof. Wolff does not take up the special question of the origin of the ore; but it would be conditioned more or less by this view. The presence and association with the ores near Mineville of the great intrusion of demonstrably igneous gabbro gives the problem regarding them some notable differences from the Hibernia ores, and, indeed, introduces another factor different from any mentioned in the other conceptions cited in a general way. As regards the hard and soft ores of Lake Superior, about whose history we have of late learned so much, it may be stated that no analogy can be traced with the present case. There is no reason to think that these have ever been siliceous carbonates of iron, which have altered to oxides and jaspers or cherts, for the latter fail, except for a rare occurrence of jasper in a cross-veinlet in the Miller Pit.

It is to be remembered that we have to deal with long, irregular bodies of magnetite, at times in double or triple beds or chutes, with much pegmatite associated, that itself contains large crystalline masses of magnetite. The ore, while extremely low in apatite in the Barton Hill group, is exceptionally high in this mineral in the Mineville group, and, perhaps, is the richest in it of any large iron-ore body in this country.* Traces of titanic acid have been met in all the mines; but it is in very small quantity and may be largely in the form of disseminations of titanite, a mineral that is common in the associated wall-rock.† On the analogy of the neighboring titaniferous ores that occur in gabbro and that are unquestionably of igneous origin—*i.e.*, by the crystallization and concentration of titaniferous magnetite in a fused and cooling magma—one might infer the same for these, lying, as they do, along the outer portions of a gabbro intrusion. But the long and relatively thin beds or chutes on Barton Hill, their double or treble

* See John Fritz in *Mineral Industry*, 1895, p. 470.

† See in this connection the analyses given in the *Tenth Census*, vol. xv., p. 111. It is a matter of regret that while the Tenth Census authorities made complete analyses of the ores from a number of the little and unimportant iron-mines of New York, none were made from those of Mineville, although the latter, from a very limited area, produced in the census year nearly half the State's total, and still rank with our largest ore-bodies.

character, and the position of the ore entirely within the acidic gneiss, as at Fisher Hill, all militate against this view.

It seems most reasonable to the writer to regard the ores here described as contact-deposits formed by the influence and stimulus of the gabbro intrusion; but, at the same time, the difficulties and uncertainties are appreciated, and the conception is supported as the *most reasonable* one, rather than as one absolutely demonstrated. Their history would then be somewhat as follows: In pre-Cambrian time, and before the complete metamorphism of the "21," Orchard and Barton gneisses from their original conditions, whatever they were, and while the rocks were deep-seated, the intrusion or intrusions of the gabbro took place. Attendant on their cooling and crystallization, the emission of highly-heated solutions of iron and other compounds transpired, presumably with vapors in some degree, and these circulating along the various contacts gave rise to the ore-bodies. They must have replaced in large part the wall-rocks and have made a place for the iron-oxide in this way. The lean ores, and there are large amounts of these at Burt Lot and in some of the drill-cores, that are chiefly quartz and magnetite, were deposited in certain localities. The coarse hornblende-magnetite-pegmatitic masses, with their associated apatites, zircons and titanites, also resulted. The irregular veins of ore of small size that are met as offshoots were formed along certian channels.* The considerable masses of fluorite surcharged with disseminated magnetite likewise resulted, and the great albite-quartz-hornblende-pegmatite, with its superb zircons in Mine 21, came into existence, although it is possible that it marked a late stage of the fumarole or "pneumato-hydatogenetic" action (if we are not unduly frightened by Prof. Vogt's formidable but significant word). By this action resulted as well the huge allanites later described and certain minor phenomena. Boracic solutions or vapors seem to have been lacking, for tourmaline has not yet been noted at the mines.

The remarkable richness in apatite shown by the red ore is esteemed a strong point in favor of this conception. Apatite veins have themselves been extremely hard problems, but the

* See, in this connection, the figure in Mr. Birkinbine's paper on the "Crystalline Magnetite in the Port Henry, N. Y., Mines," *Trans.*, xviii., 747, Fig. 2.

drift of opinion has been in late years to consider them as products of the same general processes as are set forth above for these ore-bodies. The recent papers of J. H. L. Vogt* are generally familiar and advocate this method of origin as one of the after-effects of gabbro intrusions. The richness in apatite of the Mineville ores led to their earliest development as a source of phosphates for fertilizers. In many respects the mineralogy of apatite veins and of these ores is identical.

Iron ores themselves have been referred to contact actions in Norway, and on looking the subject up, the writer discovered that the same view had been taken some years ago by Prof. Vogt for certain ores in Norway that are generally associated with granite.† Some of the sections as noted in the foot-note below are very like those of Barton Hill. Prof. Vogt even goes so far as to refer to the igneous intrusions the formation of veins as much as 1½ kilometers distant, but while one might hesitate about such remote instances as these, greater confidence may be felt regarding those at or very near the contact. As investigation of the geology of the Adirondacks progresses, it is more and more evident that the parallelism with that of Scandinavia is in many respects very close.

Regarding certain of the other mines in the region, it may be said that the Cheever ore lies in a long, forking pod a short distance above a great gabbro intrusion, but in a pyroxene gneiss. The mines at Hammondville, now abandoned, are based on a series of veins in a massive rock exactly like the Orchard gneiss of this paper, that, however, covers a considerable area and forms both walls. Great gabbro intrusions are in the neighborhood.‡

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* The most accessible statement of them for American and English readers is in the *Mineral Industry*, 1895, pp. 743–754, where references are given to other papers in German.

† J. H. L. Vogt, *On dannelse af jernmalmforekomster*, especially Part II., *Jernmalmvorkomster dannede ved pneumatolytiske processer*. ("On the Formation of Iron-Ore-Bodies." Part II., Iron-ore-bodies formed by pneumatolytic processes.) There is a *résumé* in German at the end of the paper. Fig. 12, on p. 83, is strikingly like the Barton Hill sections, except that the underlying intrusive is granite and the overlying one Silurian schist. Vogt refers to one near Vien that is associated with gabbro-diabase, p. 83. The Norwegian apatite deposits are associated with gabbros, as is well known. The above paper was published by the Norwegian Geological Survey (*Norges Geologiske Undersögelse*), Kristiania, 1892.

‡ J. F. Kemp, *Report of N. Y. State Geologist*, 1893, pp. 455–460.

Regarding contact-action elsewhere in the Adirondacks, C. H. Smyth, Jr., has recently published a very valuable paper on the zones in St. Lawrence county,* that have been such fruitful sources of finely crystallized minerals. They lie along the contacts of gabbros and crystalline limestones, and are significant in this connection as showing the presence of contact effects along gabbro intrusions rather than a similarity of degree and results. Similar views have also been expressed by the writer† for the occurrence in the Keene Valley, where iron-ore-bodies were apparently formed in crystalline limestone by the contact-effects.

Finally, as stated in the introductory remarks under this topic of origin, the writer advances this conception of an origin by contact-metamorphism not as a demonstrated thesis, but as one of strong probability and as deserving attention, and the hope is entertained that discussion of the matter may crowd the obscurity still further back.

The Mineralogy of the Mines.

The association of minerals is much the same in all the mines. The greater part of the species come from the coarsely crystalline pegmatitic aggregates, as the ores themselves supply few. Many other interesting minerals occur in quarries for limestone or other materials in the same region, although not cited here. References are made to them in the papers cited in the bibliography, especially those by H. Ries and by the writer. There is a local collection from the Mineville mines in the office of Witherbee, Sherman & Co., and one from the entire neighborhood, made by Mr. Lewis, of Moriah, is in the Sherman Academy at Moriah Center. Mr. W. H. Benedict, now of Elmira, but formerly principal of the Port Henry high-school, has collected and observed extensively in the region.

Albite.—In the coarse pegmatite of Mine 21, which affords the zircons later described, one of the chief components is a beautifully striated plagioclase of greenish hue. Masses with cleavage faces of two or three square inches may be obtained.

* C. H. Smyth, Jr., "The Genetic Relations of Certain Minerals of Northern New York," *Trans. N. Y. Acad. Science*, vol. xv., p. 260, 1896.

† J. F. Kemp, "Crystalline Limestones, Ophicalcites and Associated Schists of the Eastern Adirondacks," *Bulletin Geol. Soc. of America*, vi., 259.

The extinctions with crossed nicols on the basal cleavage are very small, ranging from parallelism to three degrees or over, and on the brachy-pinacoid, reaching fifteen. The specific gravity of a piece weighing 54.73 grammes was determined by Prof. William Hallock, at 14.5° C., to be 2.6182. This is below the average of albite (2.62), although oligoclase may in exceptional cases even reach to less values. The feldspar is therefore regarded as albite, doubtless containing oligoclase lamellæ. The general appearance is more like the latter than the former.

Allanite is the most remarkable of the minerals, viewed from the purely mineralogical or crystallographic standpoint. Itself a rather rare mineral and quite seldom seen in good crystals, it is found at Cook shaft in great abundance, of extraordinary size and in instances of great crystallographic perfection. The allanite was first announced from this locality in the *American Journal of Science*, September, 1858, p. 245, by W. P. Blake, who noted the huge crystals in the Sanford bed, the one that is now called Old Bed. Crystals, 8 or 10 inches long, 6 or 8 inches broad and ½ inch thick, are cited. These workings long since became inaccessible, although small allanites can still be found in the pit of Old Bed on the north side. Prof. James Hall secured one of the large ones in the early days, and in the *American Journal of Science*, for June, 1884, p. 479, it will be found described and figured by E. S. Dana.

At Cook shaft the huge crystals were found in great abundance on the dump, and some fine specimens were brought home in 1894 and placed in the collections of Columbia University.

Smaller and more perfect crystals have been studied by Dr. Heinrich Ries, of the mineralogical department of Columbia, at my request, and to his kindness the following identifications are due: The crystals are nearly all flattened parallel to $\infty P \bar{\infty}$, which is the most prominent face. The others found are as follows, making fourteen in all; OP, $\infty P \breve{\infty}$, $-P \bar{\infty}$, $+P \bar{\infty}$, $-\frac{1}{2}P \infty$, $+\frac{1}{2}P \infty$, $2P \bar{\infty}$, $P \breve{\infty}$, $\breve{\infty} P2$, $-P$, $+P$, $2P$. Dr. Ries intends to describe these crystals more at length.

The allanites occur in a gangue of quartz and orthoclase, and with them are rosettes of green needles supposed to be actinolite.

Amphiboles.—Dark green or black hornblende is especially

common in the coarse pegmatitic aggregates associated with the ores on Barton Hill. Where it abuts against quartz it occasionally develops the faces of the unit prism, but is as a rule only to be obtained in cleavage-pieces, which, however, are of great perfection. Actinolite is met as a microscopic mineral in the dense green rock described above among the minor rocks from the Lover's Pit.

Apatite is in enormous quantity in irregular grains, seldom over ¼ of an inch in diameter, disseminated through the so-called red ore. The red color of this apatite is due to infiltrated hematite, as in thin sections it is colorless except along cracks. In the pegmatitic aggregates of Barton Hill it forms green hexagonal prisms up to ½ an inch in diameter, and at the O'Neill shaft it was noted double this size. On Barton Hill it has been observed as an inclusion in titanite, and was evidently one of the earliest of the minerals of the pegmatites to form. The angles of the crystals are more or less rounded, and have the familiar dissolved or fused appearance.

Arsenopyrite occurs rarely in the coarse pegmatite of the lower workings of Mine 21. It is associated with quartz, orthoclase, albite and zircons. Although it occurs, as a rule, in thin seams along cracks and cleavage planes, one specimen was found by the writer, 15 mm. long, 4 mm. broad, and 3 mm. thick. Only one crystal face was developed.

Biotite is met in the coarse hornblendic aggregates on Barton Hill not infrequently. It is black and seldom in regular crystals. In the pegmatite that yields the zircons in Mine 21 it displays the long, narrow crystals peculiar to pegmatites. As a rock-making mineral it favors the rarer and exceptional gneisses that occur with the Barton Hill ores.

Calcite forms crusts of low rhombohedrons, which are $-\frac{1}{2}$ R. The Miller Pit has furnished the best, but they are rare and are products of late secondary alteration.

Feldspars.—The chief feldspars present are orthoclase and albite (oligoclase). The orthoclase is met in the pegmatites in large irregular masses, of faint pink. No crystal faces have been observed. Microcline has been noted in the Orchard gneiss, and it is possible that it is also present in the pegmatites. The albite is in large cleavage masses often showing beautifully striated faces. Its specific gravity as carefully determined for the writer by Prof. William Hallock, is 2.6182.

Fluorite, of massive character and not displaying other than cleavage-faces, occurs in some abundance in the Lover's Pit. Some years ago very pretty green and pink specimens were obtained by the writer in masses of the peculiar green rock described on an earlier page as formed of quartz and actinolite needles. The commoner white variety is thickly charged with magnetite crystals.

Garnet occurs in the North and South Pits on Barton Hill in excellent rhombic dodecahedra, which are at times distorted so as to be greatly flattened. The tetragonal trisoctahedron 202 fails of development. On Belfry Hill garnets occur in the gneisses up to 4 or 5 inches in diameter but of rough outline.

Hematite, pseudomorphic after magnetite, occurs in the pegmatite of the "21" pit. It gives the characteristic red streak but breaks up parallel with the faces of the octahedron. It is practically martite.

Hornblende.—See Amphiboles.

Hypersthene occurs as a rock-making mineral in the massive gabbro.

Jasper has been afforded by a little vein in the Miller Pit. It was in small quantity.

Lanthanite was found before 1858 by W. P. Blake in association with the large allanites earlier mentioned. It formed small crystalline plates, and probably resulted from the alteration of the allanite. (*American Journal of Science*, Sept., 1858, 245.)

Magnetite is of mineralogical interest chiefly because of the occurrence of remarkably large and perfect combinations of the octahedron and rhombic dodecahedron in the Lover's Pit in a chamber now worked out, which furnished the phenomenally rich ore referred to on an earlier page.* The crystals were buried in granular magnetite, from which they were easily broken out. They apparently showed other crystal faces; but the attempts of the writer to identify these led to the inference that they had been caused by the interference of neighboring

* John Birkinbine, "Crystalline Magnetite in the Port Henry, New York, Mines." *Trans.*, xviii., 747. The paper contains mineralogical notes by Prof. George A. Koenig.

J. F. Kemp, "Gestreifte Magnetitkrystalle aus Mineville, Lake Champlain Gebiet, Staat New York," *Zeitschrift für Krystallographie*, xix., 183. "Notes on the Minerals occurring near Port Henry, N. Y.," *Amer. Jour. Sci.*, July, 1890, 62.

faces, for extraordinary and improbable indices resulted from the calculations. Curious striations also traverse their faces, most of which follow the octahedral parting but others run parallel to still other faces.

Magnetite is plentifully found in the pegmatites, in which it exhibits, when at all regular, the familiar outlines of the rhombic dodecahedron, built up of layers parallel to the octahedron. As noted under hematite, pseudomorphs of hematite, after magnetite, but retaining the cleavage of the latter, are met in the pegmatites. Brilliant plates of magnetite, with mirror-like faces, were found at Fisher Hill by E. B. Durham some years ago and given to the writer.

In the office of Witherbee, Sherman & Co., is a remarkably perfect crystal of magnetite, possessing the symmetry of a model, and a full inch or more in diameter. It is worthy the inspection of all visitors with mineralogical tastes.

Microcline.—See under Feldspars.

Molybdenite has been found at the New Bed mines, associated with pyrrhotite. It forms the usual irregular scales, but is seldom met.

Olivine is rarely met in the massive gabbro.

Pyrite is a rarity, and is only met in some secondary veinlets. The almost total absence of sulphur in the mines is one of their striking characteristics.

Pyroxene.—Black augite occurs in the pegmatitic aggregates of the Barton Hill group and exhibits the usual eight-sided cross-sections in the prism zone. Terminal faces have not been met. As a rock-making mineral it is widespread in the gabbro and the Barton gneiss. Beautiful pyroxenes have been found by the writer on the Cheever dumps. They are described with an analysis and figured by H. Ries in his paper on "The Monoclinic Pyroxenes of New York State." (*Annals N. Y. Acad. Sci.*, ix., 151 and Plate XIV., Fig. 8, 1896.) They were bounded by the two vertical pinacoids, the prism and the unit pyramid.

Pyrrhotite in small amount has been found in New Bed, associated with molybdenite.

Quartz is occasionally met in fairly good prisms and rhombohedra of the smoky variety. Corroded bipyramidal crystals

occur in the Barton Hill pegmatites. Large masses contribute to the pegmatites in all the mines.

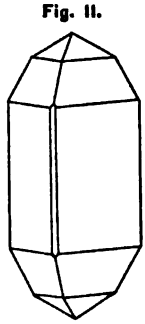
Scapolite.—See Wernerite.

Siderite is one of the minerals that helps to fill the small cross-veinlets in the Miller Pit. In the specimen gathered it formed a crust under calcite.

Titanite is found both in perfect crystals and in cleavage fragments in the hornblendic masses of the Barton Hill mines. Two types of crystals are met. In the quartz and feldspar pegmatites that are found on the dump of the Lover's Pit, light brown, doubly terminated crystals sometimes occur, of great perfection and beauty.

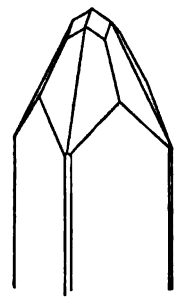
Wernerite occurs in the pegmatites of the Lover's Pit, where it was found several years ago by A. S. Eakle, to whom the

Fig. 11.



Zircon from Mine 21.

Fig. 12.



Zircon from Barton Hill Mine.

writer is indebted for a specimen. The usual combination of ∞ P, ∞ P ∞ and P formed the crystals.

Zircon occurs in two varieties. The coarse pegmatites of the Mine 21 have yielded the last year exceptionally fine crystals, of a dark brown color and up to an inch in length. They exhibit ∞ P, 3P, and P as chief faces, while ∞ P ∞ often appears as one face (Fig. 11). They have in instances undergone alteration to a greenish outside crust, much more brittle than the fresh mineral. In the hornblende of the Barton Hill mines small transparent pink zircons are met, at times doubly terminated, and bounded by ∞ P, ∞ P ∞, 3P3 and P, as is shown by the accompanying figure (Fig. 12). The determinations of the last named faces were made and the figure was drawn by the writer's friend, Prof. C. H. Smyth, Jr., of Hamilton College.

The quartz around the zircons is often filled with cracks radiating out from the zircon crystal itself. These were probably caused by the pressure through which the rocks have passed in their history, the harder zircon being able to fissure the more brittle quartz.

Bibliography.

The list given below aims only to embrace those papers that have some special bearing on Moriah township (the one containing Port Henry and Mineville) or its mines. A short bibliography of the eastern Adirondacks will be found in Kemp's *Preliminary Report on Essex County*, cited below, and in Van Hise's *Bulletin* 86, also cited below. A review of the literature is given by the former in the *Transactions of the New York Academy of Sciences*, xii., 19, 1892.

Anon.—Vacation Notes from Northern New York. On Port Henry, *Eng. and Min. Jour.*, August 31, 1889, 184.

L. C. Beck.—Report on the Mineralogy of New York State, Albany, 1842, pp. 14–16. Gives some details of the Cheever and Sanford Mines.

Sir Lowthian Bell.—Notes of a Visit to Coal and Iron Mines and Iron-Works in the United States. Read before British Iron and Steel Institute, 1875. Separate reprint, p. 21. Describes his visit to Mineville. Compare also "The Iron and Steel Institute in America," *Special Volume of Proceedings*, p. 76, 1890.

John Birkinbine.—Crystalline Magnetite in the Port Henry, N. Y., Mines. *Trans. Amer. Inst. Min. Eng.*, xviii., 747, 1890. Good account of the Lover's Pit, with notes, statistics and analyses of the ores.

W. P. Blake.—Lanthanite and Allanite in Essex County, N. Y. *Amer. Jour. Sci.*, Sept., 1858, 245.

—— Mentions Fiery Red Mica from Moriah. *Idem*, 2d series, xii., 9, 1851.

—— Contribution to the Early History of the Industry of Phosphate of Lime in the United States. *Trans. Amer. Inst. Min. Eng.*, xxi., 157, 1892. Describes early attempts to utilize the apatite of the Sanford vein.

—— Association of Apatite with Beds of Magnetite. *Idem*, xxi., 159, 1892. Advocates stratified and organic origin of apatite and magnetite.

W. P. BLAKE.—Note on the Magnetic Separation of Iron Ore at the Sanford Ore Bed, Moriah, Essex County, N. Y., in 1852. *Trans. Amer. Inst. Min. Eng.*, xxi., 378, 1892.

E. S. DANA.—On a Crystal of Allanite from Port Henry, N. Y. *Amer. Jour. Sci.*, June, 1884, 479.

EBENEZER EMMONS.—Report on the Second District of New York. Albany, 1842. Gives many geological details and notes on the mines, especially pp. 236, 237.

C. E. HALL.—Laurentian Magnetic Iron-Ore Deposits in Northern New York. *32d Annual Report N. Y. State Museum of Natural History*, 1879, 133. Gives a general sketch of Adirondack Geology and some details of the local mines.

HANNS HOEFER.—Die Kohlen- und Eisenerz-Lagerstätten Nord-Amerikas. Vienna, 1878, pp. 175–179. Plate IV., Figs. 14, 15. Gives an account of his visit and a plan and a cross-section of the ore. Regards the Mineville group as a faulted series from same original.

T. S. HUNT.—Mineralogy of the Laurentian Limestones. *21st Ann. Rep. N. Y. State Mus. Nat. Hist.*, 1871.

—— Geology of Port Henry. *Canadian Naturalist*, 2d series, x., 420. Describes the local limestones as huge veins.

—— The Iron-Ores of the United States. *Trans. Amer. Inst. Min. Eng.*, xix., 3, 1890. Refers to Lake Champlain mines.

J. F. KEMP.—Notes on the Minerals Occurring Near Port Henry, N. Y. *Amer. Jour. of Science*, July, 1890, 62.

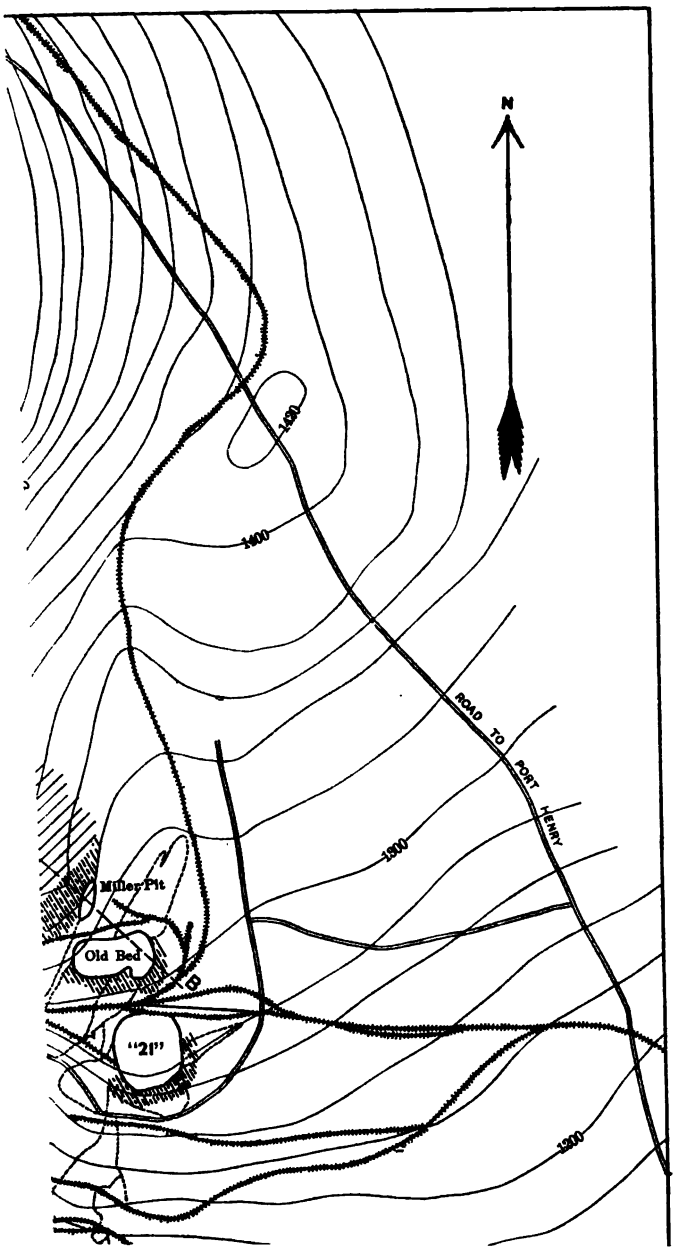
—— Gestreifte Magnetitkrystalle aus Mineville, Lake Champlain Gebiet, Staat New York. *Zeitschrift für Krystallographie*, xix., 183.

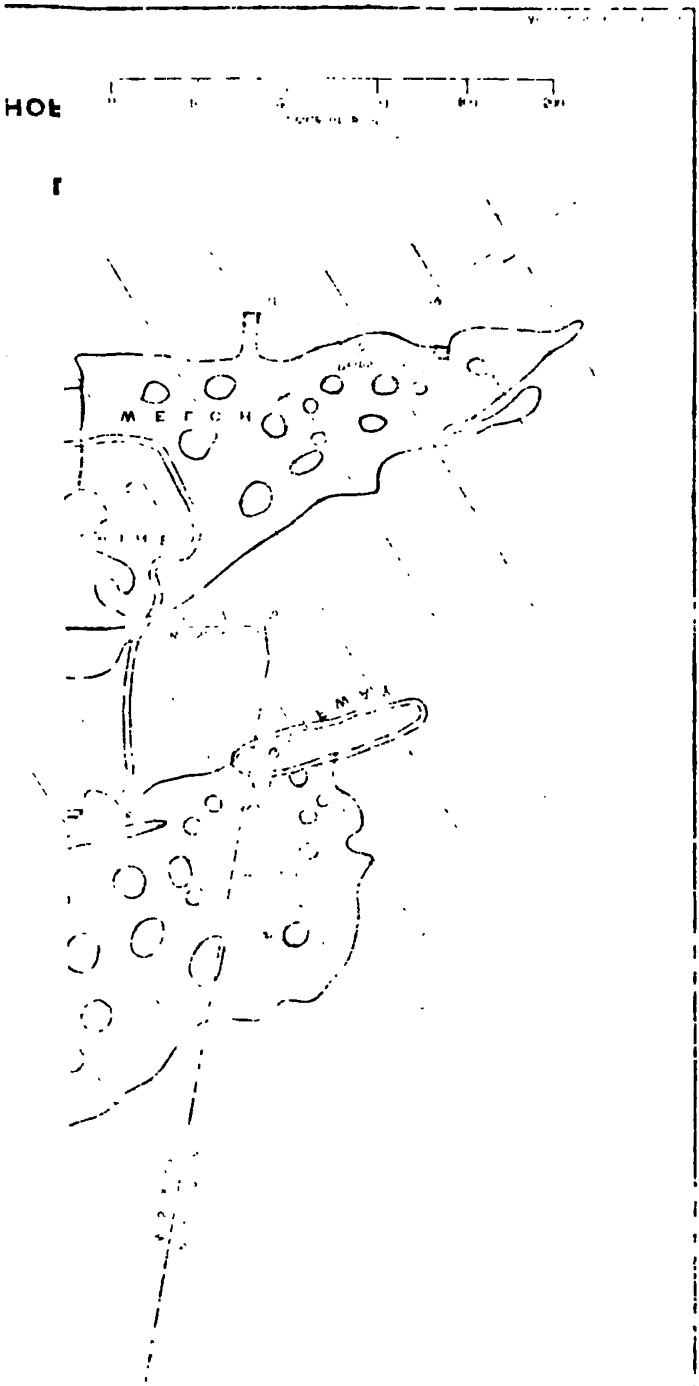
—— Geology of Moriah and Westport Townships, Essex County, N. Y., with a geological map, a map of the mines, four plates, four figures. *Bull. N. Y. State Museum*, iii., 325–355, Sept., 1895. Describes the local geology and mines.

—— Preliminary Report on the Geology of Essex County. *Report of New York State Geologist* for 1893, 433–472. Does not touch specially on Moriah township, but gives a review and bibliography of the geology of the eastern Adirondacks.

—— Gabbros on the Western Shore of Lake Champlain. *Bull. Geol. Soc. America*, v., 213. Refers to local gabbros.

—— Crystalline Limestones, Ophicalcites and Associated Schists of the eastern Adirondacks. *Idem*, vi., 241. Gives details of local geology.


N
1600
1400
ROAD TO PORT HENRY
1800
Miller Pit
Old Bed
"21"
1200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J. F. KEMP and V. F. MARSTERS.—Trap Dikes of the Lake Champlain Region. *Bulletin* 107, *U. S. Geol. Survey.* Gives some details of local trap-dikes.

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G. P. MERRILL.—On the Serpentinous Rocks of Essex County, N. Y., *Proc. U. S. National Museum*, xii., 595, 1890. Refers to the local serpentinous marbles.

F. L. NASON.—Notes on Some of the Iron-Bearing Rocks of the Adirondack Mountains. *Amer. Geologist*, xii., 25, 1893.

R. PUMPELLY discusses shape of Miller Pit, from Putnam's Notes, *Tenth Census*, xv., p. 7.

B. T. PUTNAM.—Notes on the Samples of Iron Ore Collected in New York, *Tenth Census*, xv., 89. Contains excellent details of the mines.

HEINRICH RIES.—The Monoclinic Pyroxenes of New York State. *Annals of the N. Y. Academy of Sciences*, ix., 124-180, and four plates. Gives many details of local mineralogy.

H. P. SMITH.—History of Essex County, Syracuse, 1885. The data regarding the mines are chiefly taken from *Watson's History*, which see.

J. C. SMOCK.—A Review of the Iron Mining Industry of New York for the Past Decade. *Trans. Amer. Institute of Min. Eng.*, xvii., 745, 1889. Statistical paper. See also *Idem*, xviii., 748.

—— Report on the Iron Mines of New York. *Bulletin VII.*, *N. Y. State Museum*, 1889.

C. R. VAN HISE.—Correlation Papers—Archean and Algonkian. *Bull.* 86, *U. S. Geol. Surv.*, 398. Refers to local geology.

WINSLOW C. WATSON.—History of Essex County, Albany, 1869. Gives a good historical sketch of the development of the mines.

Notes On the Northern Black Hills of South Dakota.

BY PERSIFOR FRAZER, PHILADELPHIA.

(Chicago Meeting, February, 1897.)

General Geology of the Northern Black Hills.

The "Black Hills of Dakota," to the northern part of which the following observations apply, occupy a position on either side of the boundary between Wyoming and South Dakota, approximately between 103° 15′ and 104° 48′ of west longitude and 43° 15′ to 43° 50′ of north latitude. The area is roughly 4160 square miles.* (See Plate I.)

As has been pointed out by Profs. H. O. Hofman, F. R. Carpenter and others, and illustrated by Mr. Newton in the publications of the United States Geological Survey, these hills are made up of a large number of the known formations from the Archean to the Tertiary, which latter constitutes the border of the region. The mining operations are largely confined to the interior of this area, and the centers from which they are conducted (Deadwood, Lead City, Galena, etc.) are situated near the line of junction of the Archean schists and quartzite, supposed to represent the Potsdam Sandstone of the N. Y. series, or the lowest of the Paleozoic measures.

The Archean or pre-Paleozoic rocks consist mainly of a mica-schist in which hornblende and garnet are frequently

* There are many of my fellow-members to whom the aspect of the Black Hills and the adjacent Bad Lands is familiar—in the more genial months of late spring, summer, and early fall; but to these, even, the conditions surrounding this lately re-exploited country in the middle of winter are unknown. Very lately it became necessary for me to undertake a visit of exploration to the Deadwood district while the thermometer was registering —42° Fahrenheit, and the high winds prevailing in early December of 1896 were creating havoc with those settlers who were imprudent enough to brave the elements without sufficient protection, or who had not adequately housed their stock.

By singular good luck the blizzard and I crossed each other *en route*, and I found in Northern South Dakota one of the most delightful winter-temperatures I have ever known.

found, but which varies in texture and composition so as to resemble not infrequently the hydro-mica schists or "nacrites" of the late Dr. T. Sterry Hunt, and again to assume the appearance, and in part the constitution, of heavy bedded gneiss. These schists are upturned at high, often vertical angles, while over them unconformably, and dipping gently towards the horizon is the quartzite in its various forms; sometimes apparently undisturbed, and at others broken or "brecciated," as it is often called (*i.e.*, shattered into angular fragments and recemented without very great displacement of the parts) by the numerous porphyry dikes which intersect all the exposed measures from the lowest to the highest, and form a part of the elevations or hills of the region. These latter are without question eruptive, and bear evidence of having been ejected during at least two and probably several different epochs, for I have observed porphyry in places intersecting older masses of similar porphyry, which, in their turn, have produced earlier disturbances in the strata.

Professor Carpenter's paper contains a geological section taken from the equally admirable and earlier contribution of Walter B. Devereux, on "The Occurrence of Gold in the Potsdam Formation, Black Hills, Dakota" (*Trans.*, x., 465).

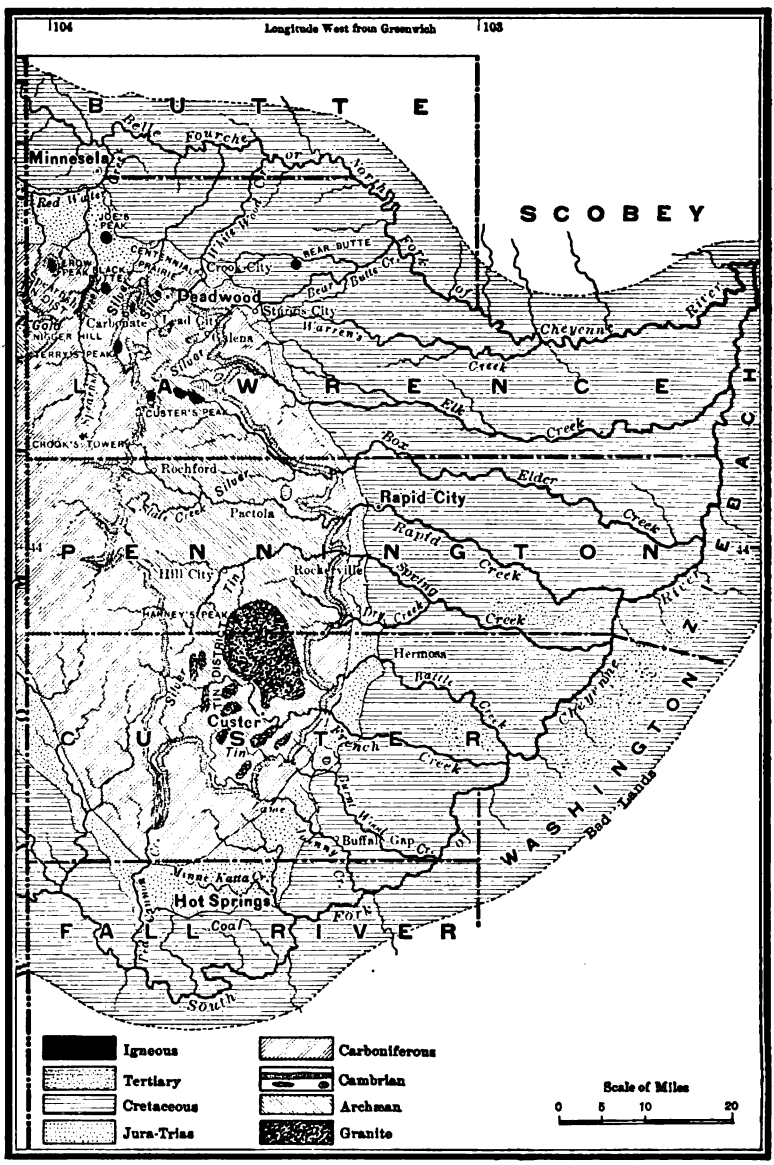
The first systematic study of this region was made by Dr. F. V. Hayden, in 1870, and resulted in the geological definition of the large divisions of the Hills much as they are accepted to-day.

Since then the Government has had the very full report on the geology and resources of the Black Hills by Henry Newton and W. P. Jenney, 1875 (Report published in 1880). Still later Devereux, Blake, Headden, Hofman, Carpenter, Rickard, Farrish, Dr. McGillicuddy and others have published valuable contributions to our knowledge of the geology, mineral resources and chemical character of the ores of the Black Hills, which will be referred to more specifically in their proper places. Plate I., on the following page, is a reproduction in black and white of the geological map published with the admirable paper of Prof. F. R. Carpenter (*Trans.*, xvii., 598) on "The Ore-Deposits of the Black Hills of Dakota."

The district which is my present subject, lies in that part of the northern Black Hills immediately around Galena and three

miles west and three miles southwest of that camp, which is some six miles southeast of Deadwood in an air-line.

PLATE I.



Geological Map of the
BLACK HILLS OF DAKOTA
After the Map of Henry Newton, E.M.
By Franklin R. Carpenter, Ph.D.

According to the topographical sheet of the United States Geological Survey (Deadwood, S. D., Ed. of Feb., 1894), the town of Galena lies on the 4800-foot contour-curve above tide. The hills along Bear Butte creek rise to 5200 and 5300 feet, making their summits 400 to 500 feet above the creek. To the northwest, or between Galena and Deadwood, and to the southwest, near the intersection of Bear Butte and Strawberry gulches, the extreme summits reach 5500 feet.

The numerous gulches follow many directions; but those which predominate are approximately northeast-southwest and northwest-southeast. The valleys are generally deep and the hillsides steep, indicating a considerable erosive action of the streams on the sides, and accounting for the numerous placer-deposits found in their beds.

The geology of the region, if not simple, is at least not complicated by a large number of formations.

The mica-schists, thought to be upper members of the Archean, are found at the lowest levels along the streams and higher up on the hills, as at the Cora mine, striking northeast-southwest, and dipping at angles from 38° to 85°.

These schists are generally micaceous and coarse-grained and usually rather reddish from the dissemination among them of iron oxides; but they vary greatly, sometimes passing into the type known as nacrite or hydro-mica schist (which here also has been frequently but incorrectly taken for talc schist) and sometimes, though more rarely, assuming a heavy bedded character reminding one of gneiss.

They are most frequently crumpled and conchoidal both in small and in large bed planes.

Explorers.—Hayden reported the Black Hills of Dakota as one of the plainest and most interesting exhibits of the series from the granite upwards, of all localities known in the West.*

Newton and Jenney issued a most instructive volume,† and one which has served as a mine for all future observers. The map accompanying this work is that generally used for all purposes of geological illustration of the region to-day.

* Meek and Hayden: "Some Remarks on the Geology of the Black Hills and Portions of the Surrounding Country," *Proc. Acad. Nat. Sci., Phila.*, vol. x., pp. 41-49, 1859.

† "Report on the Geology and Resources of the Black Hills of Dakota," *U. S. Geol. Survey*, 1880.

Walter B. Devereux* thought that the oldest visible formations near Deadwood were paleozoic schists (?).

Resting unconformably upon them was a sedimentary formation composed of débris derived from them, *i.e.*, the Potsdam. The lowest is a conglomerate. The mines in the lower part of the quartzite are called cement-mines. The deposits were found on the east side of the Homestake vein. Only one deposit was worked on the western side.

The mass of the conglomerates was a mixture of quartz boulders, pebbles and schist, with pebbles of hematite; gold was present as a mechanical, and in the cement-beds as a chemical, constituent. In general only 5 or 6 feet of rock will pay for mining or milling. Cutting through a dike, he found that gold was seldom present in close proximity to the porphyry. Gold plates are found along the interior plane of the "talc" (?) schist. The schists underlying this deposit were found to contain gold for about 10 feet. The ores of Bald mountain seem to be impregnations.

1. The metallic constituents were segregated along certain horizontal strata. 2. There was concentration along certain vertical planes which were contacts of quartzite with porphyry dikes. The dikes themselves were found to be lines of mineral segregation below the contact of quartzite and of the schist. Devereux inclined to the belief that the gold came partly from below and partly from the concentrated sediments above.

In one place the sedimentary stratum gave $20 per ton in gold, though so fine that no gold could be panned.

Argentiferous lead-ores have been mined from the quartzite. These mineralizations probably are of the age of the porphyry.

Dr. Headden† thought that the strike of the granites was the same as the schists with which they are interstratified. The granites further north are more varied. It is a question whether they are continuous undeground. On the eastern margin of the schists occur lenses, some large and some small, with their longer axes parallel. Many of the granites are intrusive. There is no porphyry in the southern section of the Hills.

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* *Op. cit.*, *Trans.*, x., 465, Feb., 1882.

† "Notes upon the History of the Discovery and Occurrence of the Tin-ore in the Black Hills," by Dr. W. P. Headden, *Proc. Colo. Sc. Soc.*, vol. iii., pt. 3, p. 347.

The Cambrian is unconformable on the schists, part of which form its basal conglomerate.

Prof. Hofman* says that the gold is found in quartzite and pyrite finely divided through mica- and amphibole-schists, and impregnates the schists. Its associated minerals are pyrite, with some arsenopyrite, chalcopyrite, garnet, and asbestos. The ores from the upper cuts and levels are more free-milling than those below the water-line. The assay-value of pure concentrates is $4 to $90 per ton; the average perhaps $25 per ton. The porphyry forms barren horses.

From the valuable contribution to this subject made by Prof. Carpenter† I have compiled the following summary:

The Cambrian is represented only by the Potsdam and rests unconformably on the Archean schists. No Silurian or Devonian is found, and the Potsdam and Carboniferous are conformable. The great deposits are of Archean age and yield mainly auriferous pyrite. There are sometimes lenticular and seemingly independent masses of the slate and schist series sharing the folds and contortions and having a columnar cleavage coincident with the bedding. The porphyry flowed over the Potsdam east of Deadwood and under that in the Homestake to the west, lifting it like a laccolite. The gold was deposited by chemical action after having been segregated.

The smaller the particles the finer the gold (Devereux).

Assays of Homestake porphyry show that it contains no gold. The whole thickness of the beds in which these ore-bodies occur is about 2000 feet.

The ore-bearing rocks are co-extensive with the igneous rocks.

The Potsdam ores occur at different levels, especially in the upper parts of the lower quartzite. Bluish quartzite contains very finely divided pyrite in a siliceous paste, which unites the rounded grains of silica. The solutions which brought this material to fill the spaces between the grains brought also the silver and gold. At Galena the proportion of gold is greater in the lower contact, while silver predominates in the upper, which are the more calcareous parts of the Potsdam. In the limestone the deposits carry lead and silver; but the porphyry is the same. The solutions followed certain bed-planes which are on the quartzite and in the lime-shales of the Potsdam. The connection between the overlying porphyry and the ore is intimate. Where one is wanting, so also is the other. The crystalline schists below have nearly always shown gold. Perhaps the porphyry only induced hot waters which dissolved and redistributed it. The slates below the Potsdam at Galena show galenite also (in the Oro Fino mine). Blende, galenite and pyrite occur in the porphyry.

At Galena there are three distinct "contacts" or levels of ore, always on quartzite, on which it is found replacing parts of the thin lime shales, and over it, though not always in contact, is porphyry.

No ore-bodies have been found to pass from the porphyry into the Archean.

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* "Gold-Milling in the Black Hills," by H. O. Hofman, *Trans.*, xvii., 498, Feb., 1889.

† *Op. cit.*, *Trans.*, xvii., 570, Feb., 1889.

The outcrop of the Galena deposits is almost always lead and iron carbonates carrying silver. The depth at which these become sulphides varies with the position of the deposit; in the lower contact, quickly; in upper contacts, they continue to greater depth.

The "veins of granite" of Newton, Jenney, Blake, and Vincent are segregated parallel to the apparent bedding. They are lens shaped and succeed each other in strike, and probably also in dip. They are tabular in form (?). When tin is present, one of the constituents of the granite is wanting, and the rock becomes quartz and mica, or quartz, feldspar and mica.

The quartzite which is the main locus of the ores lies unconformably upon these schists, not deviating more than 20°, but usually presenting the appearance of being in general nearly horizontal This quartzite is sometimes banded, and so compact as scarcely to differ from a quartz-vein aspect; but usually it is of comparatively uniform light color, interspersed here and there with small crystals of pyrite and arsenopyrite, and, in the region under discussion, much broken from the intrusion of numerous dikes of porphyry.

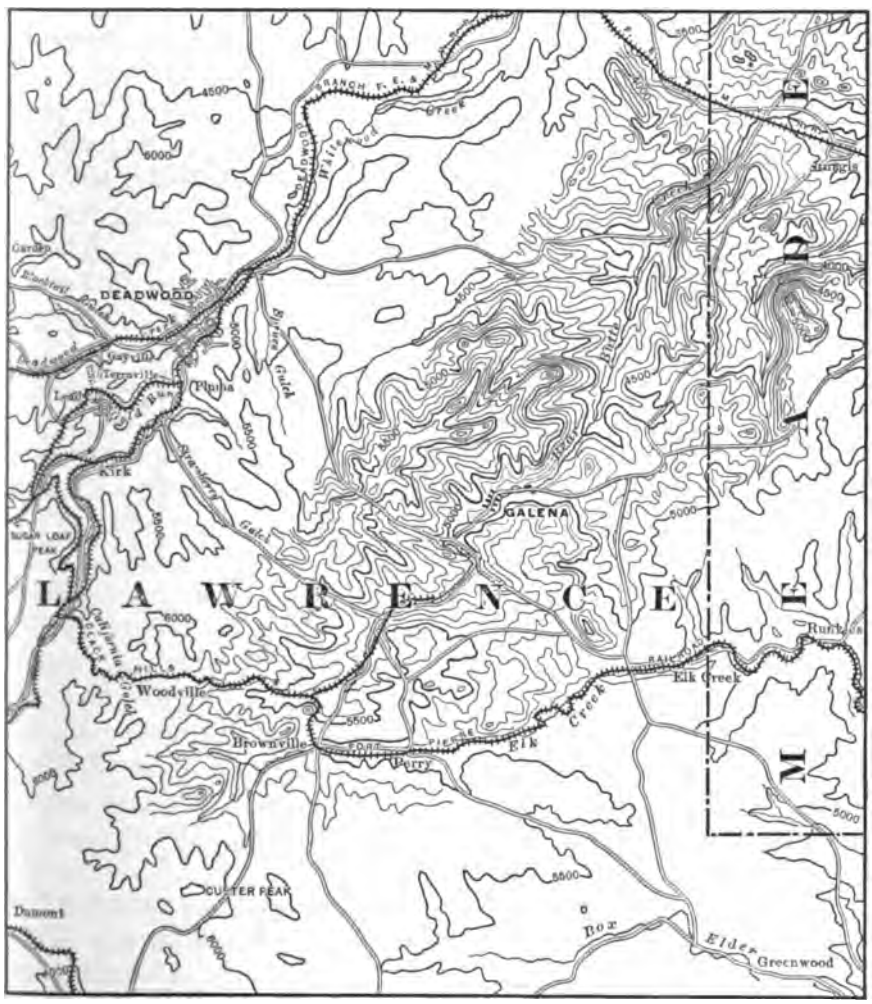
Galena, zincblende, arsenopyrite and other accessory minerals are found on the fracture-planes of the quartzite, and to a certain extent within its mass near to these planes. There are three levels of the quartzite rock in what has been taken for the Potsdam formation, and there is evidence of quartzite horizons.

In general it may be said that the alteration of the original sandstone to quartzite is more complete in the lower than in the higher horizons, and that the highest of these siliceous members of the Potsdam is capped by a series of shales or "lime-shales."

The topography of the Galena region has already been stated in general terms. (See Plate II.) It consists of a network of narrow and deep gulches separated by steep hills whose shoulders are about 300 feet, and whose crests are 500 feet above water-level. It is more than probable that the directions of these narrow valleys in many instances indicate the directions of porphyry outflows between the broken quartzite and shale, since the decomposed porphyry is the most easily eroded of all the rocks in the region, and would offer the least resistance to the wearing action of running water. Still, other causes were potent in modeling the hills as we now see them, among which a very important one must have been the agency of moving ice. However this may be, the steep portions of the hills up to the shoulders just spoken of are largely composed of quartzite. The summits, in those localities where the porphyry has overflowed, show by their gentle slopes the characteristic moulding of weathered porphyry; but in other localities a different contour is noticeable, characteristic of the almost equally soft "lime shales." It is important to keep in mind this general distribution of the porphyry, and especially so when considering the Union Hill mining properties (See Plate III.),

because the Hill is largely composed of porphyry, and, from the state of the gold which it contains, constitutes a valuable ore. The earlier geologists who examined the Black Hills were

PLATE II.



GALENA DISTRICT, SOUTH DAKOTA.

(Reduced from Deadwood Sheet U.S.G.S.)

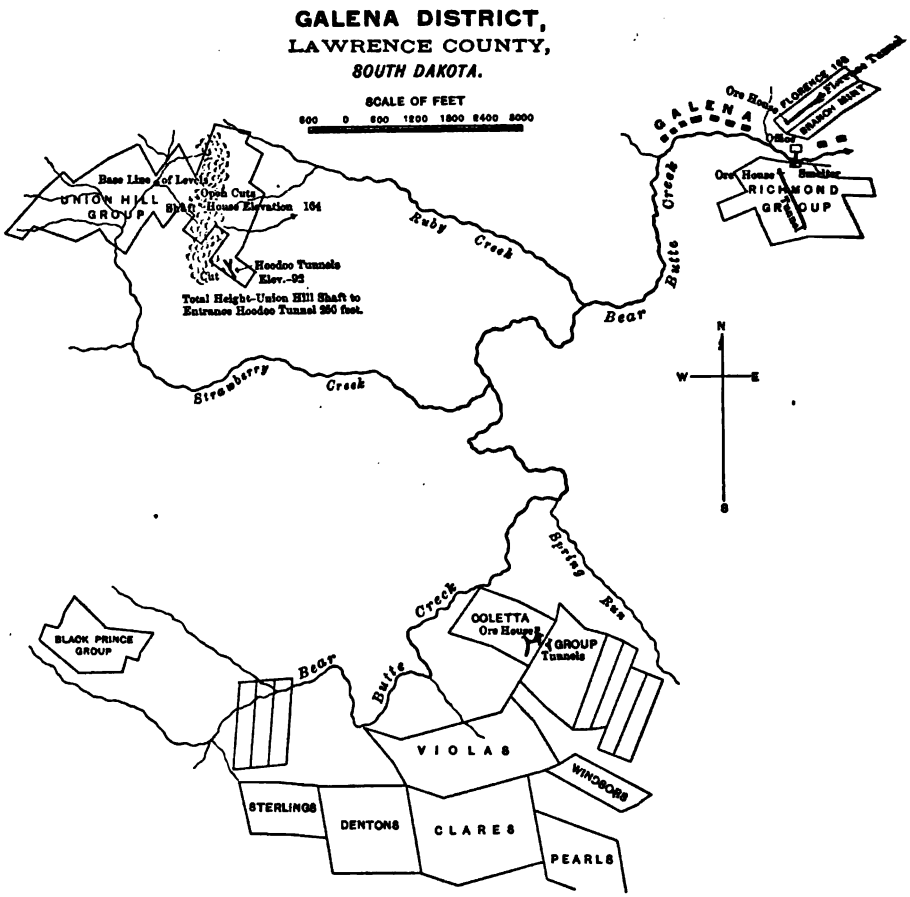
Scale of Miles

1 0 1 2 3 4 5

of the opinion that this rock was barren of valuable metals. Thus Devereux reported that about 5 or 6 feet of the quartzite would pay for mining and milling, and that "gold was seldom

present in close proximity to the porphyry." He thought that the planes of contact, fracture and bedding were enriched, and that 10 feet or so of the schists below the quartzite (the schists underlying the Potsdam) contained gold. The Bald mountain

PLATE III.



deposit he regarded as an impregnation. Carpenter believed that the Homestake porphyry contained no gold, but that the thickness of the beds in which ore-bodies occur was 2000 feet, and that the *ore-bearing* rocks were co-extensive with the igneous rocks. He was of the opinion also that the solutions which brought the finely-divided pyrites to fill the spaces between the grains, brought silver and gold; that the connection

with the porphyry was intimate in the Galena region; but that, perhaps, the porphyry only induced the hot waters which dissolved and redistributed the gold.

Blende, galenite and pyrite, he says, occur also in the porphyry, *but no ore-bodies have been found to pass from the porphyry into the Archean.*

Prof. Hofman mentions the quartzite and schists as carriers of gold, but describes the porphyry as forming "barren horses."

Nevertheless, subsequent tests of at least a part of the porphyry outflows show that, to some extent, free-milling gold is carried in this rock, and on account of its enormous extent and the ease and economy with which it can be mined, it is important to determine its richer zones. To this end a number of openings have been made on the surface of Union Hill. A shaft nearly 300 feet deep to the bottom of the sump has been sunk, and a tunnel of 359 feet N.W. from mouth to heading has been run, with a cross-cut 123 feet E. of N., begun from a point 227 feet from the mouth of the main tunnel.

Starting at a cabin stable on Strawberry creek, near Mr. Fagan's house, I ascended the hill in an easterly direction, taking carefully-averaged samples from a pit at the base of the hill. The material is a weathered porphyry and the pit about 10 feet deep (nearly filled with snow at the time of my visit). Between this pit and the open cut, about 150 feet N.E. and considerably higher, nothing was observed on the surface but porphyry in various stages of decomposition. The rock in which it is excavated is also a rotten porphyry, of which large masses are exposed. In this cut a small natural bridge of rock has been left standing.

About 50 feet west of this cut is another, also in decayed porphyry, in which the principal cleavage-planes dip more or less westerly. Both the last-mentioned openings are near the shaft-house at the top of the hill.

The shaft-house covers a shaft 280 feet deep, with a sump of about 20 feet below the second level. At the time of my visit the lower level was inaccessible, and I visited only the first, or 140-foot, level, from which two drifts had been started, one extending 30 feet S.E. and the other 40 feet N.W. of the shaft.

At the breast of the S.E. drift a "horse" of "barren quartzitic

rock" had come in at the roof, inclining E. about 45°, and apparently of lenticular shape.

The porphyry in the N.W. drift is remarkable for being very full of disseminated pyrite. It is said to be the only instance of the kind thus far found in the Union Hill shaft.

In a cut near the shaft-house a loose mass of quartzite was observed, lying in porphyry.

Renewing the examination of the surface of the hill from the top of the shaft, I visited a cut in white porphyry about 700 feet east by south of the latter and obtained samples for analysis. An experimental shaft in porphyry, about 200 feet S. 10° E. of the last and 180 feet lower, also furnished samples which were analyzed. A winding open cut in rock, similar to that which has been above described, about 50 feet long, about 250 feet south of the last point, and said to be about 50 feet above the floor of the main Hoodoo tunnel, was examined and sampled. Finally, a short tunnel in rotten porphyry, with apparently horizontal cleavage-planes, about 150 feet W. 20° N. of the mouth of the main Hoodoo tunnel, yielded its quota to the material collected for investigation.

About 150 feet E. 20° S. from this cut is the mouth of the Hoodoo tunnel proper. The opening of this tunnel is in porphyry like that of the other openings described, but with planes of cleavage (thought by some to correspond to the stratification of the disturbed quartzite), dipping S.E. 20°. These planes are also noticeable in the upper and shorter tunnel.

This rotten porphyry extends inward for about 60 feet, where it is replaced by a broken quartzite mixed with fragments of slaty material. This character is maintained for another 64 feet (or to a point 124 feet from the mouth) where a heavy-bedded flaggy slate is met on the hanging-wall, dipping 40°—S. 35° W. No porphyry, however, is visible. On the foot-wall is the "brecciated" slate, with fragments of quartz, some of it banded. Between this and the spot just referred to were two places some 50 feet apart, where brecciated slates and quartz formed the principal part of the rock. At the heading of the Hoodoo tunnel, 859 feet from the mouth, occurs a fragmental quartzite following more or less the plane of bedding of the slates. According to report, the quartzite mass here is about 40 feet below the lower surface of the por-

phyry outflow. This under surface must, therefore, cut the tunnel at 60 feet from its mouth, and its slope would be 1 foot in 7 or 8 to the south.

The upper Hoodoo tunnel is about 15 feet higher in level than the main tunnel, and is all in porphyry. If this be the main body of porphyry and possess an even under-surface, it would make the slope more nearly 1 in 2, and imply a dip of 30° from the mouth of this tunnel to the intersection of the porphyry and the main tunnel.

In the cross-cut from its junction with the main drift for 63 feet, the rock is fragmental quartzite interspersed with schists and shales; but at this point the porphyry appears and continues to the heading.

Assuming the floor of the upper tunnel to be fifteen feet higher than that of the main tunnel and the distance 150 feet to the intersection of the porphyry and the cross-cut, the fall of the latter would be 1 in 10, corresponding to an angle of about 6°. If these two bodies of porphyry are connected, as is practically certain, the facts would indicate that the strike of this sheet of porphyry is N.W. and S.E., and that the hill west of Strawberry creek and the Union Hill proper are on the same porphyry dike.

This hill is distant from the shaft-mouth about a quarter of a mile, on the west side of Strawberry gulch. The surface of the hill shows the same decomposed porphyry as Union Hill, above described, and the shallow openings, wherever made, bear out these indications; but it is not certain how thick this cap of porphyry may be. On the summit of the hill and about a quarter of a mile in a direct line from the Union Hill shaft is a cut in decomposed porphyry 30 feet long, 17 feet deep and 2½ feet wide, in the bottom of which a nearly vertical vein widens to 2 feet from 8 inches at the surface. Small crystals of galenite are said to have been found in this vein. It is also reported that galenite has been taken from a dark ferruginous gangue.

Near by is a shaft 126 feet deep. At the deepest point, an iron-ore is said to have been found, of which specimens were seen on the dump-pile. On both sides of the shaft appears a vein, which, at the depth of 100 feet, is said to have been 5 feet wide.

A very instructive and interesting phenomenon, of a dike of

porphyry cutting porphyry, was observed near the cut on top of this hill. It has a nearly vertical dip and a strike of N. 35 W.

If this observation be correct, it proves that there were successive outflows of porphyry at different epochs, and it may well be that one of these (not improbably the later) was more barren of the precious metals than the original outflow. At least a difference in age would confirm the hypothesis that there are different kinds of dikes in this region, some of them auriferous and some of them entirely barren. One of the latter is seen on top of the hill near the Cora mine, S.W. of Galena.

The assays of the specimens taken were as follows:

No.	Description.	Dollars per Ton.		Total Dollars per Ton.
		Gold.	Silver.	
1	Surface of Union Hill	0.4134	.....	0.4134
2	" "	0.8268	0.096	0.9228
3	" "	0.4134	0.096	0.5094
4	Average ore, experimental pit, 700 feet E. by S. of U. H. shaft	Traces.	.....	.....
5	Do., experimental pit S. of U. H. shaft	.....	.....	.....
6	Do., deep cut, 165 feet N.W. of mouth of Hoodoo tunnel	11.57	0.228	11.7980
7	Do., deep cut	3.7206	0.480	4.2006
8	Shaft, Union Hill, "Horse," first level, heading S.E. drift	0.4134	.....	0.4134
9	Do., first level, heading S.E. drift	0.4134	0.096	0.5094
10	Do., first level, heading N.W. drift	0.4134	0.072	0.4854
11	Upper Hoodoo tunnel	0.4134	0.070	0.4834
12	Main Hoodoo tunnel	3.6139	0.192	3.7059
	Total gold and silver in 12 tons of samples.	.....	.....	$23.41
	Average per ton	.....	.....	$1.95

Two samples collected by Mr. David Jones in the Hoodoo tunnel last October and assayed by different persons gave $43.70 and $4 per ton respectively.

Messrs. Merrill and Sandt collected two samples of ore from Union Hill last September which gave $8.56 and $4.70 per ton, respectively, or a mean of $6.63.

No sample of ore collected in the field can give as just a view of the real value of an ore-property as the results of a mill-run; and I am therefore much indebted to the courtesy of Captain

Boyd and the officers of the company in permitting me to use the following data obtained from a mill-run undertaken last October. This mill-run, as he explains, was of Union Hill ore, but was made during unusually cold weather, which froze the water on the plates in spite of all precautions. Despite these unfavorable conditions, he obtained $117 in gold from 40 tons of ore, which gives a value of $2.92 per ton. (He estimates that the unfavorable conditions of the run resulted in a loss of nearly two-thirds of the gold or $5, which would make the value of the ore treated $8 per ton, exclusive of the value of the concentrates, which he thinks will represent 1 to 80 in the porphyry ore of Union Hill, and 1 to 5 in the ore of the Hoodoo tunnel.)

The available quantity of the Union Hill ore depends upon the proportion which the porphyry of the hill bears to the country-rock above water-level. What we know about the hill is that its outer shell is composed of porphyry, and a part of its inner nucleus of the quartzites and shales, through which the former has forced its way. Various views have been held in regard to the geological epoch and the manner in which these outflows took place.

Numerous evidences of flow along the bedding and between the strata, as well as through rifts made by the upheaval of the measures, may be found in this part of the Black Hills. A similar rock in Wales has been thought by eminent geologists to be the result of volcanic outburst through several separated vents followed by a shower of ashes, which latter were later compacted into felsitic beds, which in course of consolidation become porphyritic. It is quite probable that there have been many such vents in this region, though several of them probably united in the production of a single sheet of molten rock.

Various observations tend to support the view that the porphyry of Union Hill strikes N.W. and S.E., and may have been derived either from an outflow through a fracture in that direction (since this does not correspond with the strike of the Archean schists) or from a series of vents, produced between the planes of the schist at different horizons, which united their outflows into one sheet.

The broken and fragmental character of the rock in parts of the Hoodoo main and branch tunnels would indicate the close

proximity of the cause of disturbance, even if the actual intersection of both these drifts by porphyry were not observed. On the other hand, the continuance of porphyry (as I am reliably informed) throughout the 300 feet of the shaft, indicates either that the shaft has accidentally been located in a vent, or that the whole covering of porphyry is prodigiously thick. The former seems the more probable supposition, because, at a short distance from the shaft, a large mass of quartzite may be seen embedded in the igneous rock, and apparently at a considerable distance from any quartzite in place, as if it had been carried bodily upward in the igneous flow.

The specific gravity of an average specimen of the porphyry was found to be 2.755. A cubic foot of this rock would thus weigh 171.94 pounds, and a cubic yard about 2⅓ tons. The distance on the Union Hill property from the line of Strawberry gulch and its small eastern branch to the mouth of the Hoodoo tunnel is about 2418 feet. Assuming a width of 1000 feet, which would include a large part of the company's property, and 256 feet, as the surveyor's estimate of the vertical depth of the mouth of the Hoodoo tunnel below the shaft-house, we have 619,008,000 cubic feet. Taking half of this amount, or 309,504,000 cubic feet, as the cubic contents of the hill, and half of these contents, or 154,752,000 cubic feet, as consisting of porphyry, there would then be 26,617,344,000 pounds, or 13,308,672 tons of porphyry above water-level. Assuming the average value of $1.95 per ton which resulted from the assays of ore which I selected, this would represent $25,951,910. Assuming the value actually obtained by Captain Boyd's mill-run of 40 tons (and without allowing for the loss which he believes to have taken place) or $2.92 per ton, the contents of the Union Hill would be valued at $38,861,322.

I add a table, which, from a professional point of view, must be regarded as a curiosity. It represents assays of twelve samples of carefully averaged ore made by three assayers of high reputation. There is no doubt whatever that the material of each of these triplicate assays was in a metallurgical sense the same.

For the purpose of getting a better insight into the true character of these porphyries, several specimens collected at different parts of Union Hill were subjected to examinations under the

Union Hill Ores, near Galena, S. D.

Mark.	Assayers.	Gold.		Silver.		Total Value of Gold and Silver.
		Ounces per Ton of 2000 Pounds.	Value, $20.67 per Ounce.	Ounces per Ton of 2000 Pounds.	Value, 60 Cents per Ounce.	
1	A.	0.100	$2.067	.....	.....	$2.067
	B.	0.300	6.200	.....	.....	6.200
	C.	0.230	4.754	2.35	$1.41	6.164
2	A.	0.08	$1.6536	.....	.....	$1.65
	B.	0.15	3.10	.....	.....	3.10
	C.	0.08	1.6536	1.08	$0.64	2.2936
3	A.	0.10	$2.067	.....	.....	$2.067
	B.	0.20	4.130	.....	.....	4.130
	C.	0.13	2.68	0.89	$0.53	3.21
4	A.	0.06	$1.2402	.....	.....	$1.24
	B.	0.25	5.1700	.....	.....	5.17
	C.	0.14	2.8900	0.46	$0.276	3.166
5	A.	0.08	$1.6536	.....	.....	$1.6536
	B.	0.125	2.58	.....	.....	2.58
	C.	0.08	1.6536	0.90	$0.54	2.1936
6	A.	0.06	$1.2402	.....	.....	$1.2402
	B.	0.125	2.58	.....	.....	2.58
	C.	0.14	2.89	0.81	$0.486	3.376
7	A.	0.04	$0.8268	.....	.....	$0.8268
	B.	0.125	2.58	.....	.....	2.58
	C.	0.04	0.80	0.36	$0.216	1.016
8	A.	.....	.....	.....	.....	.....
	B.	0.125	$2.58	.....	.....	$2.58
	C.	0.03	0.6201	0.31	$0.1860	0.8061
9	A.	0.02	$0.4134	.....	.....	$0.4134
	B.	0.125	2.58	.....	.....	2.58
	C.	0.05	1.035	0.25	$0.150	1.185
10	A.	.....	.....	.....	.....	.....
	B.	0.1	$2.06	.....	.....	$2.06
	C.	0.08	1.65	0.20	$0.12	1.77
11	A.	0.04	$0.8268	.....	.....	$0.8268
	B.	0.075	1.55	.....	.....	1.55
	C.	0.04	0.8268	0.40	$0.24	1.0668
12	A.	0.040	$0.8268	.....	.....	$0.8268
	B.	0.200	4.13	.....	.....	4.13
	C.	0.07	1.4469	0.37	$0.224	1.6709

Union Hill Ores, near Galena, S. D.—Continued.

	Assayers.	Gold.	Silver.	Total Value of Gold and Silver.
		Value, $20.67 per Ounce.	Value, 60 Cents per Ounce.	
Average.	A.	$1.0676	.....	$1.0676
	B.	3.27	.....	3.27
	C.	1.9083	$0.4181	2.3233

microscope as coarse powder and as thin slides in polarized light. Three specimens of this coarse powder were also kindly examined by Dr. Edgar F. Smith of the University of Pennsylvania, in his laboratory, for their percentage of alkalies and of silica.

Microscopical Examination of the Concentrates from the Porphyry of Union Hill.

It was an interesting question, in connection with this free-milling rock, in what state the gold occurred, whether free or attached to the pyrite or other minerals. The condition of the silver, which the assays proved to exist in eight or ten times larger quantity than the gold, was also very interesting.

An experienced assayer reported the existence of "concentrates with the appearance of pyrites largely mixed with hard particles of hematite, which was probably derived from the decomposition of the pyrites. A few colors of gold were visible which have a good yellow color. . . . . In all of the samples the relation of gold and silver fluctuates with some regularity."

(L).—An examination of a quantity of concentrates under a magnification of about 100 diameters revealed gold in the free state and of a good yellow color (showing it unalloyed with silver), but in almost every case attached to a pyrite fragment.

In several instances the association was as a lining to a little cavity, or in a crack of the pyrite, differing from it in color and luster and apparently uncontaminated by combination with any other substance.

Of the other metallic minerals present in the concentrates, pyrite, hematite and some limonite (named in the order of

their quantity) formed perhaps 90 per cent. Besides these there were numerous non-metallic mineral fragments, both of original and of secondary character, which were very interesting and unexpected. It should be remembered that the greater number of the specimens of porphyry from which the concentrates were obtained came from points on or near surfaces which had been exposed to moisture and air for a considerable time, and therefore cannot be supposed to represent with certainty the character of the interior mass of the porphyry.

Following roughly the order of their frequency, the minerals were orthoclase (both pegmatolite and sanidine), pyroxene, and a yellow silicate(?), the nature of which was not determined with certainty, but which is not suspected of being either rare or important. It may be serpentine.

A second examination of this concentrate follows:

A comparatively large amount of the porphyry was concentrated by panning, and examined under magnifying powers of 60 and 120 diameters, and proved to contain pyrite, quartz, rounded milky grains of feldspar, also reddish translucent grains with occasional circular groups on a surface exhibiting high refracting power, and resembling pegmatolite. Occasionally these two appearances are exhibited by the same grain. Translucent and generally rounded grains showing pale bluish or bluish green color and opalescence in transmitted light were observed; also yellowish, barely translucent grains, and a minute mass of the finest hair-gold interwoven into little knobs, and spongy in appearance.

There were also fragments of a dark bluish black mineral with metallic lustre and perfect cleavage on at least one plane (galenite) and argentite(?), diopside, beryl (one fragment), and cassiterite(? ?).

Dr. Amos P. Brown, of the University of Pennsylvania, gives as his description of this powder: (1) Magnetite, (2) pyrrhotite, (3) pyrite, (4) chalcopyrite, (5) beryl, (6) serpentine(?—the yellow material alluded to above), (7) cassiterite(? ?), (8) quartz in small amount.

(9. V. 21).—Another specimen was concentrated, and gave the writer pyrite, quartz, pegmatolite, yellowish translucent silicate, native gold, and galenite.

Dr. Brown says of this: "It seems to have the same con-

stituents as (L), but with a large percentage of the magnetic portion, which is mainly pyrrhotite, it would seem." He adds: "As I have no data as to the method of separation of their concentrates, I do not know anything of the specific gravity, and can hence form but a poor guess of the constituents. The dark minerals are not in crystals, so the color and lustre only can be used in determination."

A mass of this same powder which had not been concentrated was given to Dr. Edgar F. Smith, from whom I have the following partial determination:

	Per cent.
SiO_2,	63.53
Al_2O_3 and Fe_2O_3,	22.89
K_2O and Na_2O,	7.35
CaO,	2.80
MgO,	not determined
Undetermined,	3.63
	100.20

The rock is therefore a rhyolite with numerous mineral inclusions, including native gold and combined silver, some of which are, in all probability, of secondary origin, and a few of which may have been transported from other places, inasmuch as all the specimens were taken at no great distance from the surface, and the rock was very generally broken and weathered.

An analysis of this rock was made by Mr. V. Lenher in the John Harrison laboratory of chemistry of the University of Pennsylvania. It proved to be composed of:

	Per cent.
SiO_2,	61.79
Al_2O_3,	17.79
Fe_2O_3,	4.01
CaO,	0.80
MgO,	0.23
K_2O,	4.78
Na_2O,	7.56
Loss on ignition,	3.46
	100.42

This rock resembles in most of the constituents, except the silica and alkalies, a trachyte from Ischia near Naples; but in these latter percentages is nearer to an acmite-trachyte from the Crazy mountains, Montana; both cited by Prof. Kemp.

(1. V. 9).—The examination of this rock in coarse powder gave much the same results as that of the last, except that the

grains, though smaller, were less translucent, and no free gold was observed.

Dr. Brown describes it as containing: "(1) Magnetite, (2) pyrrhotite, (3) pyrite, (4) diopside (? small amount), (5) quartz and flint. The same dark mineral as in (L) may be (6) cassiterite. (7) Some globular and botryoidal grains closely resemble some forms of cassiterite (or uraninite). There is also a little red opal. Possibly (7) may be limonite or psilomelane, but it is rather black." An analysis of this rock, unconcentrated, made by Mr. George William Sargent, in the Harrison chemical laboratory of the University of Pennsylvania, yielded the following results:

	Per cent.
SiO_2,	51.93
Al_2O_3 and Fe_2O_3,	31.35
Na_2O,	2.89
K_2O,	6.275
CaO,	0.75
MgO,	0.51
Loss from ignition,	7.26
Total,	100.965

The silica in the specimen is very low, and the alumina and iron sesquioxide are very high; yet its position with reference to other more normal specimens leaves no doubt that it is part of the same outflow, modified, no doubt, by weathering.

(E).—This porphyry was obtained from a shallow pit about 900 feet E. of S. of the shaft-house on Union Hill, and but a short distance above the main Hoodoo tunnel.

It yielded to the examination of its concentrated powder very much the same results as those above. Quartz and feldspar predominated, and the pyrite was smaller in amount. The quantity of this concentrate was less than the others, though made from an equal quantity of powder. A small fragment of hematite was observed.

Dr. Brown says of it: "Much like (10 V. 22), a large part of the magnetic part seems to be pyrrhotite. Quartz, barite(?), not much pyrite; much black mineral, which looks like rutile or cassiterite."*

* A considerable quantity of this powder was examined for me by Dr. S. P. Sadtler for the presence of tin, with negative results. The existence of cassiterite in this rock must therefore be considered as yet unproven.

The unconcentrated powder of this rock was subjected to a partial analysis under the direction of Dr. Smith, with the following results:

	Per cent.
SiO_2,	66.41
Al_2O_3 and Fe_2O_3,	18.63
CaO,	2.50
Undetermined,	12.46
	100.00

A specimen of the unground rock from this pit was cut and mounted in thin slides for examination by the polarizing microscope.

Under 60 and 120 diameters it proved to consist of a fine granular and hyaline matrix, including a considerable amount of quartz and secondary quartz, in which were imbedded large crystals of some sanidine and pegmatolite. A small amount of mica was also suspected.

In another specimen the sanidine showed considerable corrosion of the edges, and the matrix was of a texture suggesting the presence of silica deposited from solution.

This rock is also a rhyolite, less rich, or perhaps entirely destitute of valuable metallic constituents.

Dr. Brown describes this section thus: "Sanidine in porphyritic crystals, some micro-pegmatitic. Grains of primary quartz and much secondary quartz. This rock had apparently a glassy matrix originally. I do not find muscovite."

In a later note he says: "It is a rhyolite without doubt. I should say it had originally a rather glassy base, but this had become devitrified. It is not apparently a very acid rhyolite, however."

An analysis by Mr. V. Lenher in the John Harrison laboratory of chemistry of the University of Pennsylvania demonstrated:

	Per cent.
SiO_2,	67.10
Al_2O_3,	13.95
Fe_2O_3,	2.72
CaO,	1.54
MgO,	0.59
K_2O,	8.31
Na_2O,	2.43
Loss on ignition,	3.85
	100.49

From its silica, alumina, iron sesquioxide, lime and magnesia, this rock resembles one from Flagstaff, Colo.; and in the alkalies a rhyolite from Silver Cliff, Colo., of which analyses are published in the *Handbook of Rocks.*

(10 V. 22).—The appearance under the microscope of the concentrated powder resembles very much that of (9. V. 21), but it contained less pyrite, and no gold was observed.

Dr. Brown's description is as follows: "(1) Magnetite, (2) pyrrhotite, (3) quartz, (4) pyrite, (5) same material as (7) in (1. V. 9), (6) also some of the dark mineral of (L. 7), and yellow of (L. 6), perhaps serpentine. Much quartz, and a larger proportion of pyrrhotite than in the others."

An unconcentrated specimen of this rock was analyzed in the laboratory of the University of Pennsylvania by Mr. George William Sargent, and gave the following results:

	Per cent.
SiO_2,	58.71
Al_2O_3 and Fe_2O_3,	25.65
Na_2O,	3.61
K_2O,	6.69
MgO,	trace.
CaO,	0.29
MnO,	0.73
Loss from ignition,	4.20
Total,	99.88

The silica is still rather low for a rhyolite, which, nevertheless, we must suppose this to have been in its original state.

A specimen of a coarse-grained rock (*a*) obtained from a point near the Union Hill shaft on the surface of the hill, when cut into thin slides and examined under the microscope, exhibited a fine-grained matrix, in which rested short, thick slabs of labradorite and large prisms with pyramidal terminations of much disintegrated augite. Besides these, are some crystals of biotite, also small grains of some undetermined silicates and a little secondary quartz. The peculiar fragmentary condition of the pyroxene is striking. It is as if in pseudomorphs, or what appear to be such, though the original form may have been pyroxene.

Much of the granular matrix, associated with the usual accessory minerals accompanying augite or labradorite, is made up of small augite fragments.

This rock seems to be an augite-andesite.

Dr. Brown says of the sections of this rock: "It contains plagioclase, a little sanidine, augite (light-colored), and in a^2 (one of the slides) a little brown hornblende. Very little if any secondary quartz, augite-andesite."

He adds in a subsequent note: "I think this could be safely called a pyroxene-andesite or augite-andesite, although I am of the opinion that some of the pyroxene may have been rhombic. 'Pyroxene-andesite' would be safe."

The Coletta Group.

The Coletta group of mines lies nearly due east of the Black Prince, 1⅓ miles down the valley of the Bear Butte creek. The openings are situated on the top of a steep hill which separates Spring run from Bear Butte creek, and at an average of 300 feet above the beds of the streams. The rock is a quartzite, reported by Capt. Boyd to be here 175 feet thick and overlying the Archean schists, which are nearly vertical. The dip is accordingly slight, about 5°—E. by S., and the ore conforms to this dip, but occurs in a peculiar form, called by Capt. Boyd "corrugations" or "corrugates." From whatever cause, there have been deposited on the upper surface of the quartzite uneven lenticular masses of ore like sausages strung together by slender strings, and dipping longitudinally towards the east. They lie approximately in the same plane at intervals of 30 to 40 feet, parallel with each other, and are intersected by other similar masses at right angles, which increase both the richness and quantity of the ore at their junction.

The cause of this strange phenomenon has been thought to be furrows or troughs made in the surface of the quartzite in which the metalliferous solutions deposited their contents; but the origin of the depressions, and especially of these gridiron-like forms, is not clear. The length to which they continue is not known, the exploitation being terminated in most instances by the contour of the hill.

Measurements of these corrugations show that they are broad and shallow lenticular masses, from 5 to 12 feet wide and from 8 inches to 3 feet thick. Where these lens-shaped masses are united end to end, or at the parts analogous to the connecting-string of the sausages, the rock becomes sandy, and free

gold is found. The mineral matter consists mainly of carbonates of lead containing 50 cents to $12 per ton in silver and 0.2 ounce to 1½ ounces of gold to the ton, and varying between 1 and 40 per cent. of lead. These masses are said to increase in value with depth, which is natural from the manner of their deposit; the heavier and richer metals and their compounds having drifted to the lower planes.

They are all more or less ferruginous, and siderite and magnetite are reported to have been found in them.

The ore exposed is a carbonate of lead containing nodules of cerussite.

Tunnel No. 1, overlooking the valley and the hamlet called Virginia City, was opened on the nose of the hill by Mr. Harris in 1891, in a small shaft or pit sunk by an earlier prospector.

The first pipe, encountered about 4 feet from the tunnel-mouth and drifted on, was 1 foot wide, and increased shortly to 4 feet. The tunnel runs about S. by E., and the drift about E. The greatest thickness of these corrugations was 4 feet, and their greatest width 20 feet. Here the dip of the rock was 5°—E. by S. in a decomposed quartzite. About 150 feet from the mouth of the drift it turns and is run on the small pipe. There are other horizons in the quartzite where the lenticular and gridiron-structure occurs, and pipes of ore connect them.

A number of these tunnels are opened along the northern and eastern face of the hill at not very different levels.

On the eastern face of the hill, in opening No. 5, the plane of the pipes has a strike of about S. 35° W., and the pipes themselves dip about 15°—E. 35° S.

Samples from all the tunnels, Nos. 1 to 5, were taken and averaged, the product being indicated by the letter J below.

A nodule of bright red color was selected from No. 5 and assayed. Vanadinite was found in parts of the face, and especially in small crevices. Specimens were taken from opening No. 4 for examination. At the mouth of No. 5 a pipe 18 inches thick, of yellow color, and containing heavy lead carbonates, was observed. A short distance into the tunnel a flat lenticular mass or pipe, 10 feet wide and of unknown length, runs into the hill at an angle of about 15°—N. 35° W. Some 30 feet south is another twin pipe or corrugation connected with it. This last has a breadth of about 4 feet where observed.

About 100 feet S.W. from No. 5 is another opening, No. 6, into another pipe. Little was found outside these openings as dump, which is indirect testimony favorable to the character of the material excavated.

At 100 feet E. of opening No. 6, and 25 feet lower, a large cavern or stope has been excavated under the lower edge of the quartzite, where a quantity of kaolin appears, and a considerable quantity of ore. The horizon is thought by Capt. Boyd to be the lower edge of the Potsdam, and it is not unlikely that it marks at least the lower edge of a lower Potsdam quartzite. A banded quartzite called "ribbon-quartz" runs through the Archean schists at many localities in this country, which is said to be usually gold-bearing.

At the Lynn, or No. 7, cut, 250 feet from the mouth and at the heading, a series of pipes was found with carbonates of lead averaging from 170 to 500 ounces of silver. This ore was sacked. (About 30 sacks make a ton.) Four tons or more have been taken from this hole. Specimens taken from both sides of this drift were averaged and sampled, and the assay of the resultant specimen, labeled L, will be seen below. (Vanadinite was also found here.)

The mouth of the Lynn, or No. 7, tunnel is about 300 feet above water-level, and the extreme summit of the hill (which lies about S.E. of tunnel No. 1, and a little north of west of the Lynn tunnel) is about 50 feet above it, but Capt. Boyd estimates the average cover of the mines at about 20 feet.

In general, it may be remarked that in tunnel No. 1 and the openings nearest to it, the pipes or corrugations are in rotten disintegrated rock; but to the southeastward the quartzite in which the pipes are found becomes harder and firmer.

About 300 feet S. 25° E. of the Lynn, or No. 7, tunnel, is another trial-shaft which showed some ore. At a depth of about 8 feet, at the bottom of the shaft, a pipe is found striking N. 25° W. and S. 25° E., but the ore is under such light cover that a great deal of it has been disseminated among the clays.

According to Mr. Lynn, the surface indications of a pipe are a red soil and float-ore.

The nearest Archean schists underlying this quartzite which were observed, dipped S. 35° E.—85°.

The assays of the specimens taken from the Coletta workings here follow:

Assays from Coletta Workings.

Mark.	Description.	Dollars per ton. Gold.	Silver.	Lead. p. c.	Dollars, gold, silver and lead per ton. (Lead at 3.25 c. per lb.)
J	Tunnels 1 to 5, . .	7.4412	1.008	5.6	12.0892
Ja	Bright vermilion nodule in No. 5, . .	29.3514	10.068	——	39.4194
K	Tunnel No. 5, . . .	0.4134	0.096	11.17	7.7694
L	Lynn or No. 7 tunnel, .	2.0670	27.408	3.67	31.860

The lack of continuity in the ore-sheet caused by the breaks in the spaces between the gridiron of ore-bodies does not prevent a reliable estimate of the amount of ore from being made, for these corrugations persist with remarkable regularity. It was not easy to make a test of the amount of free-milling gold which occurs at the points where the ore-bodies pinch out, but it is likely to prove a large addition to the value of the ore here given. The lead is a valuable associate, and facilitates greatly the treatment in the smelter. With improved means of transportation to the Galena plant, which has already been projected, a large, steady and profitable output may be confidently relied on.

In the upper part of Galena, near the angle of Bear Butte creek referred to, the Archean schists dip S. 30° E.—50°. The exposures show the rock in the hill to be compact, and in places curved and foliated. At this place a contact with porphyry is not far off; in fact nature seems to have arranged the greatest possible number of contacts in this entire region, and to have prepared each contact with its quota of ores. The realization of this fact will explain the great prevalence of the word "contact," which one hears constantly.

At the tunnel in the Cora mine, previously referred to, the opening is at the bottom of a middle quartzite dipping about 5°—S.E. At the upper tunnel, at the junction of the Archean schists with this quartzite, is plainly shown the former group, a pinkish mica schist dipping E. 20° S.—38°. (This low dip is doubtless due to the sagging of the laminæ exposed on the upper side of the hill.) The quartzite lying immediately upon it dips E. 10° S.—18°.

Mr. Mall, an old and experienced miner, states that in general, the quartzite of this region is thought by the experienced miners to dip E. at a low angle, and the schists S.E.

It has been mentioned that on top of the Cora hill is a capping of porphyry destitute of valuable mineral contents. The cleavage-planes of this rock seem to conform to bed-planes of the quartzite and dip 60°—E. 10° S.

At another opening on a contact of quartzite with the overlying shales, a quantity of cherty-looking red oxide of iron is observable. Lamellar oxides of iron with conchoidal fracture and smooth surfaces follow this upper contact. It would be important to know whether this merges into pyrite deeper and further into the mine, where it is less exposed to the weather.

The Cora mine produced galenite, silver and gold.

List of Papers on Economic and Structural Geology of Black Hills.

I am indebted to Mr. Walcott, the Director of the United States Geological Survey, for the following bibliography on the economical and structural geology of the Black Hills:

Englemann. Report of a geological exploration from Fort Leavenworth to Bryan's Pass made in connection with the survey of a road from Fort Riley to Bridger's Pass under command of Lieutenant F. T. Bryan. *Rept. of Sec'y of War*, 1856. Appendix H, p. 508.

Meek and Hayden. (Some remarks on the geology of the Black Hills and portions of the surrounding country.) *Phila. Acad. Sci., Proc.*, vol. x., pp. 41–49, map. 1859.

Winchell (N. H.). Reconnaissance of the Black Hills of Dakota, U. S. A. *Rept. of Chief of Engineers*, 1875, Part 2, Appendix PP, pp. 1131–1172.

Newton (Henry). Geology of the Black Hills. Introduction. *Report on the Black Hills of Dakota*, J. W. Powell in charge. Washington, 1880.

Jenney (Walter P.). Mineral resources of the Black Hills of Dakota. *Rept. on the Black Hills of Dakota*, J. W. Powell in charge. Washington, 1880.

Devereux (Walter B). The occurrence of gold in the Potsdam formation, Black Hills, S. Dak. *Trans. A. I. M. E.*, vol. x., 1882, pp. 465–475 (with section near Homestake).

Newberry (J. S.). The Deposition of Ores. *School of Mines Quart.*, vol. v., p. 336.

Blake (Wm. P.). Tin-Ore Veins in the Black Hills of Dakota. *Trans. A. I. M. E.*, vol. xiii., pp. 691–696. 1885.

Chance (H. M.). Resources of the Black Hills and Big Horn Country, Wyoming. *Trans. A. I. M. E.*, vol. xix., p. 49.

Carpenter (Franklin R.). Notes on the Geology of the Black Hills. *Preliminary Report of the Dakota School of Mines upon the Geology, Mineral Resources and Mills of the Black Hills of Dakota*, pp. 11–52, map. 1888.

The Mineral Resources of the Black Hills, their Character, Occurrence, and Extent. *Preliminary Report of the Dakota School of Mines, etc.*, pp. 107–171. 1888.

Ore-deposits of the Black Hills of Dakota. *Trans. A. I. M. E.*, vol. xvii., pp. 570–593. 1889. (With colored geological map.)

Crosby (W. O.). Geology of the Black Hills of Dakota. *Bost. Soc. Nat. Hist., Proc.*, vol. xxiii., pp. 488 517, 1888. See also vol. xxiv., p. 11.

Van Hise, (C R.). The Pre-Cambrian Rocks of the Black Hills. *Geol. Soc. Am., Bull.*, vol. i., pp. 203-244. 1890.

Ward (Lester F.). The Cretaceous Rim of the Black Hills. *Jour. of Geol.*, vol. ii., pp. 250–266.

Posepny (F.). The Genesis of Ore-Deposits. *Trans. A. I. M. E.*, vol xxiii., pp. 197–369.

Todd (James E.). A Preliminary Report on the Geology of South Dakota. *S. D. Geol. Surv.*, Bull. No. 1, pp. 172.

Pirrson (L. V.). On Some Phonolitic Rocks from the Black Hills. *Am. Jour. Sci.*, 3d ser., vol. xlvii., pp. 341–346.

Hall (Charles E.). A Vacation-Trip into the Black Hills of South Dakota. Abstract: *Minn. Acad. Nat. Sci., Bull.*, vol. iii., pp. 185–186.

Hall (W. S.) The South Dakota Artesian Basin. *Science*, vol. xxii., pp. 29–30.

Headden (Wm. P.). Stannite and some of its Alteration Products from the Black Hills, South Dakota. *Am. Jour. Sci.*, 3d ser., vol. xlv., pp. 105–110.

Claypole (E. W.)., The Tin-Islands of the Northwest. *Am. Geol.*, vol. ix., pp. 228–236.

Ulke (Titus). A New Tin-Mineral in the Black Hills. *Trans. A. I. M. E.*, vol. xxi., pp. 240–241.

A Contribution to the Geology of the Dakota Tin-Mines. *Eng. and Min. Jour.*, vol. liii., p. 547.

Rolker (Charles M.). The Production of Tin in Various Parts of the World. *U. S. Geol. Surv.*, 16*th Ann. Rep.*, Part III., pp. 488–538.

Todd (J. E.). Recent Geological Work in South Dakota. *Am. Geol.*, vol. xvi., p. 202.

Rickard (T. A.). Variations in the Milling of Gold-Ores—XII. The Black Hills, South Dakota. *Eng. and Min. Jour.*, vol. lx., pp. 221-223 and 247-251.

See also *Bulletin* No. 127, *U. S. Geol. Surv.*, Catalogue and Index of Contributions to *N. Amer. Geol.*, 1732-1891, by N. H. Darton, under the heading Dakota.

Biographical Notice of Joseph D. Weeks.

BY ALFRED E. HUNT, PITTSBURGH, PA.

(Chicago Meeting, February, 1897.)

By the death of Joseph Dame Weeks, past-President of this Institute, which occurred December 26, 1896, the world has lost an earnest and unwearied philanthropist; the Christian church, a zealous, active and consistent representative; the United States and the State of Pennsylvania, a patriotic, pub-

lic-spirited and unselfishly useful citizen; social and technical science, an esteemed authority; the American Institute of Mining Engineers, an early member, three times an officer, a valued contributor to its *Transactions* and always an efficient supporter of its prosperity; while hosts of friends have lost, so far as earth and time are concerned, that most precious of possessions, a true friend.

Mr. Weeks was born in 1840 at Lowell, Mass., and, in his early manhood, was employed as a chemist in the laboratory of Dr. J. C. Ayer, a manufacturer of proprietary medicines in that city. Later, he became a student at Wesleyan University, with the view of entering the ministry; but, in addition to his theological studies, he paid special attention to the physical sciences, to which, by the natural bent of his mind, he was specially attracted. After his graduation he was for eight years a Methodist preacher in New England, and although in subsequent years the chief work of his life was connected with sciences, industries and professions not distinctively religious, he maintained throughout the devout and consecrated spirit, the strong sense of duty and the lofty ideal of service which had inspired his earliest choice of vocation.

During the civil war he became one of the most efficient agents of that noble and beneficent institution, the United States Christian Commission, which, by its labors for the material and moral interests of the soldiers of the Union, contributed, not less truly than did the hosts actually in arms, to the welfare of the nation, while mitigating the horrors of the conflict. In this capacity he was able to trace the fate of many wounded soldiers or prisoners of war, and of many who had died in hospitals or on the field, thus bringing to innumerable homes the satisfaction (sad or joyful, as the case might be) of a certainty which removed the element of anxious doubt, worse even than grief.

After the close of the war he lived for some three years in Cincinnati. It was perhaps during this period that he turned his attention to journalism, a profession for which his varied experience had qualified him. For some time he was a writer for the Burlington *Hawkeye*, a local Iowa newspaper with a more than local reputation.

In 1872 he removed to Pittsburgh, Pa., where he continued to

reside until his death, earning by his character and work, as the years went on, the position of one of the most widely known and highly honored citizens of that important place. His first work at Pittsburgh was in the editorial department of the *American Manufacturer and Iron World*—a position in which his early scientific training was fruitfully helpful.

Early in 1875 Mr Weeks left the Pittsburgh journal and became an associate editor of the *Iron Age* of New York—a position which he retained for some ten years.

But the rapidly increasing predominance of Pittsburgh as a center of the iron and steel industry of the United States opened a field for local technical journalism not to be neglected; and in 1886, severing his connection with the *Iron Age*, Mr. Weeks became, and remained until his death, the chief editor and the manager of the *American Manufacturer and Iron World.*

The management of such an enterprise was enough for one man, even of exceptional capacity for work; but the incomplete outline of the activities of Mr. Weeks, given below, exhibits an amazing industry beyond the limits of selfish business interests. The secret of his multifarious activity lay largely in the circumstance that he was universally known as one who would never refuse to undertake a labor which he deemed, on the one hand, honorable, and, on the other hand, useful to Pittsburgh, or the United States, or Christianity, or science, or mankind. It is impossible to present a complete record of his indefatigable service under all these heads, and even the incomplete account given below is not arranged in accurate chronological order, but rather, in a general way, in the order of the spheres of usefulness just named.

As a representative of the manufacturing interests of Pittsburgh, Mr. Weeks appeared, in connection with tariff questions, before many Congressional Committees, commanding respect by his known conscientious honesty and his thorough knowledge, based on painstaking personal research, of the subject at issue. He was one of the founders, and at the time of his death the secretary, of the Western Pig Iron Association. He was also one of the founders of the Western Iron Association. He was prominently connected with the iron industry at the time of the formation of the great labor-union known as the Amalgamated Association of Iron and Steel Workers of the

United States, and personally edited and presented the first "scale of wages" offered by the manufacturers to that powerful organization. Yet although generally recognized as the spokesman of the manufacturers, he enjoyed to an unusual degree the confidence of the representatives of "labor." It was the Pittsburgh coal-miners who first proposed him to preside over a board of arbitration between them and their employers. He often served as the chairman of "wages-conferences," and succeeded on many occasions in the reconciliation of conflicting interests and demands. The deep regret occasioned by his death in the circles of "organized labor" has been eloquently expressed in a tribute to his memory published by President Garland, of the Amalgamated Association of Iron and Steel Workers. It is indeed unfortunate that such tributes are so rare; but, on the other hand, it is matter for congratulation and hope that the example of Joseph D. Weeks has demonstrated the value and power, in industrial controversies, of sincere good-will, simple honesty and fair play.

Mr. Weeks was one of the founders, and, at the time of his death, a director, of the Pittsburgh Chamber of Commerce.

He was also one of the founders of the Homœopathic Hospital of Pittsburgh, and a trustee of that institution for many years, during which he saw it become, and helped to make it, one of the largest and best-appointed hospitals in Pennsylvania.

Moreover, he was a member of the Press Club, the Duquesne Club and the Junta Club (a literary association), all of Pittsburgh. If to these data we add that no enterprise for the benefit of Pittsburgh or its industries lacked his hearty cooperation, it will be clear that he was a citizen of conspicuous loyalty and devotion, and the universal recognition of him as a representative of that city will need no explanation.

Yet he was not less active in promoting the interests of the whole country. Perhaps more widely known as an intelligent and careful statistician than in any other capacity, he was repeatedly employed by the United States Government in the preparation of its census reports and the annual reports of the Geological Survey on the mineral resources of the country. The State of Pennsylvania has also received, from time to time, the benefit of similar service from him. In his special field of

the industries of iron, steel, coal and coke, petroleum and natural gas, he took rank as the highest authority. At the Chicago Columbian Exposition of 1893, his eminent ability was recognized in his appointment as one of the Judges of Award in the Department of Mines and Mining. It may be remarked, in passing, that the hard work which he performed in this capacity broke him down physically, and proved to be, as now appears, the beginning of the end of his earthly activity. He never fully recovered from the illness which followed it, and his sudden death from apoplexy was but the sequence of several physical warnings which he had received.

Notwithstanding these warnings, he continued in harness to the last. At the time of his death he was burdened, in addition to the regular editorial and other labors connected with his journal, with the duties of Chairman of the Pennsylvania Tax Conference Commission, a body charged by the State Legislature with the grave responsibility of submitting to the State Department at Harrisburg a revised general revenue bill. Into this work he had thrown himself with characteristic zeal and industry, and had already collected a vast amount of material for use in the wise formulation of revised tax-laws for the State.

As if this were not enough, he was at the same time engaged in gathering information for the guidance of the National Commission appointed to appraise the property of the Monongahela River Navigation Co., under an Act of Congress authorizing the Government to purchase the dams and improvements owned by that company, and to make the navigation of the Monongahela free.

His interest in the progress of science, especially in its application to industry, was keen, intelligent and far-seeing. He was one of the first to recognize and to urge upon the attention of manufacturers the advantages of natural gas as fuel. To his persistent and energetic advocacy the tin-plate manufacture of this country may almost be said to owe its existence and present success. The year before his death he visited Great Britain and the continent of Europe to investigate the progress of the utilization of by-products from special coke-ovens, and returned imbued with the conviction that the coke-manufacturers of western Pennsylvania and Virginia were suffering large

and unnecessary losses through the dissipation into the atmosphere, from bee-hive ovens, of the valuable volatile constituents of the coal. Of course, it is not claimed for him that he was the first or the only one to call attention to this subject, or that the particular method with which he identified himself is the only method for effecting the desirable end he had in view. But it is certain that the general interest in the matter has been largely increased since the publication of his reports upon foreign practice.

Mr. Weeks became, in 1875, a member of the American Institute of Mining Engineers, with which, it is needless to say, his connection was not that of a passive recipient of its benefits. His frequent attendance and hearty co-operation at the meetings, and his contributions to the *Transactions* (enumerated below), won for him a deserved prominence. In 1886 and 1887 he served as Vice-President; in 1890, 1891 and 1892 he was a Manager; and in 1895 he became President.

The files of the two technical journals with which he was associated contain a vast amount of his work which cannot here be catalogued in detail. Outside of this, the following list of essays and reports, though doubtless incomplete, will indicate the nature and amount of his literary activity.

In the Reports of the U. S. Census of 1880: Articles on "The Manufacture of Coke," "The Manufacture of Glass," "Statistics of Wages in Manufacturing Industries," and supplemental reports (in Volume XX.) on "Strikes and Lock-Outs," and "Trades Societies."

In the Reports of the Department of Mineral Resources of the U. S. Geological Survey: In each volume from the one giving the statistics of 1885 to the latest, published in 1896, articles on "Crude Petroleum," "Natural Gas," and "Manganese."

In the Reports of the U. S. Census of 1890: For the Department of Mines and Mining, articles on "Crude Petroleum," "Natural Gas," and "Manganese;" and for the Department of Manufactures, articles on "Coke," "Glass," and "Petroleum-Refining."

In the Transactions of this Institute: "Hadfield's Patent Manganese-Steel" (xiii., 233); "Tests of Manganese-Steel (xv., 461); Remarks on the Kerosene Blowpipe (xiii., 678); Remarks on Relief Associations (xii., 605); Remarks on

American Blast-furnaces (xix., 984); Remarks on the Consumption of Natural Gas (xx., 411); Biographical Notice of William Powell Shinn (xxi., 394); "The Elk Garden and Upper Potomac Coal-Fields of West Virginia" (xxiv., 351); "Some Fuel-Problems" (Presidential address, xxv., 943); and a Presidential address delivered at the Pittsburgh meeting, February, 1896, which has not yet been issued officially by the Institute. This address dealt with the claim of William Kelly, a native of Pittsburgh, as the first to conceive the principle of the Bessemer process. I am informed by the Secretary that the delay in its publication was due to the express order of Mr. Weeks, who was not willing to have it issued in the form in which, through what he deemed a misconstruction of his meaning, it had been bitterly criticized by the friends of Sir Henry Bessemer. He intended, therefore, to revise it in such a way as to remove all possible misunderstanding. But this intention, I believe, he had not executed when overtaken by death.

Mr. Weeks was an earnest worker in the Methodist Episcopal Church, for many years superintendent of a Sunday-school in Pittsburgh, and one of the founders of the Valley Camp Meeting Association of Western Pennsylvania. He was well known as an earnest advocate of temperance, and personally a total abstainer from both alcoholic liquors and tobacco. In politics he was an ardent Republican, and served as the Secretary of the Republican National Committee in the campaign of 1892. Always ready to work for his party, he was never an office-seeker.

He was a Free Mason, of the thirty-second degree of the Scottish rite.

He was one of the founders of the Order of the Knights of Pythias, of which he wrote the only history in existence.

At the time of his death he was a member of the Board of Management of the Sons of the American Revolution of the State of Pennsylvania. He came of good Revolutionary stock, his great-grandfather, Jacob Heard, having been a soldier in the regiment of New Hampshire militia which fought at Bunker Hill, and afterwards, reorganized as a regiment of the regular Continental army, continued in service to the end of the war.

A noble and beautiful feature of his character was the unfail-

ing interest and sympathy with which he regarded young men starting out in the field with which he was familiar. During the last twenty years Pittsburgh has called to aid in the development of her great industries many young men of scientific tastes and training; and the writer speaks for himself, as well as for a multitude of others, in recalling with grateful acknowledgment the warm greeting received at Mr. Weeks's hands, and the ready willingness with which he furnished at all times technical or statistical information or friendly advice, free from all airs of assumed superiority, and full of bright, hearty fellowship and good-will.

In 1871 Mr. Weeks married Miss Mattie J. Fowler, daughter of a gentleman once prominently connected with the Pittsburgh iron industry. His widow and one daughter survive him.

Biographical Notice of Alexander Trippel.

BY R. W. RAYMOND, NEW YORK CITY.

(Chicago Meeting, February, 1897.)

MOST of us, in this hurrying age, take little pains to preserve such records of our doings as will make the work of our biographers easy. Now and then there is an exception, and Dr. Trippel was one. Methodical in all his habits, he kept diaries and journals comprising his own personal experiences, the details and results of his numerous scientific investigations, and his impressions concerning men and events; and also copies of all his technical essays, published by various societies or periodicals. It is a striking instance of "the irony of fate," that these carefully-preserved materials, which would now have a real historical, as well as biographical value, were totally destroyed by an accidental fire a few years ago.

In addition to my own recollections of an acquaintance covering more than thirty years, I am indebted, for the facts upon which this sketch is based, chiefly to Mr. Eugene J. Trippel, a son of the deceased, and now Register of the United States Land Office at Tucson, Arizona. Mr. Trippel, however, frankly confesses that his own knowledge of his father's work is, in the absence of written records, imperfect.

Alexander Trippel was born January 25, 1827, at Schaffhausen, Switzerland. One of his family, in an earlier generation (though not a direct ancestor), was the celebrated sculptor, Alexander Trippel, whose bust of his intimate friend, Goethe, is said to have been characterized by Bayard Taylor as the most perfect work of sculpture extant. It is not unlikely that he was named after this eminent artist; but if this was the case, and if any hope was thereby symbolized that the bearer of the name would become a disciple of art, the hope was a delusion. The boy developed, in the schools of his native town, a special affection and aptitude for mineralogy, geology, chemistry and the kindred physical sciences. In truth, the rapid and even romantic development of these branches, in the first half of the present century, gave them an irresistible attraction for studious minds; and, no doubt, many an ardent young spirit, which might have become, a generation earlier, a devotee of poetry, literature, scholastic learning or art, was drawn to the pursuit of science. Moreover, it was at that period that science began notably to "pay;" so that even a prudent parent might well consent to let his son adopt that path. At all events, this particular father, a sturdy citizen of Schaffhausen, and himself possessed of no mean talents and accomplishments, was satisfied to hold his only son closely to his studies, whatever they were. He was a stern taskmaster, and the boy got little time to play; but this was not a specially distasteful restriction. In boyhood, as in later years, he found his recreation in his work.

His professional education as a chemist and metallurgist was completed in Germany—probably at Freiberg, though I have not been able to establish the fact; and there is also a tradition that he studied at Heidelberg, where, perhaps, he took his degree as Doctor of Philosophy—the department of "philosophy," as is well known, including all branches of knowledge not comprised under jurisprudence, medicine or theology.

He came to this country, I believe, somewhere about 1848, and, at the earliest period permitted by our laws, became a citizen of the United States, and remained throughout his life a loyal and ardent American.

My own acquaintance with him dates from about 1863, but others can recall his activity at earlier dates. Thus, Mr. Charles J. Eames, of New York, writes that he remembers Dr. Trippel

in 1857 as a consulting expert in that city, engaged in metallurgical researches, especially upon copper-ores. His New York office, and the consulting business carried on in connection with it, he maintained for a considerable period, notwithstanding frequent, and sometimes prolonged, absences. It was still his headquarters when I knew him first. For some time he had as associate the Belgian chemist and geologist, Prof. Eugene Gaussoin; but while he was still alone in business, it was his custom, when called away on professional work, to close his office without ceremony or warning, leaving a card upon the door, with the legend, "Gone to South America" (or some other remote locality); "will soon return!" In such a case, "soon" might mean a month or a year.

It is, of course, impracticable to give, in the absence of contemporaneous records, a consecutive account of his activity. Perhaps his first important work was the erection of sulphur-works at Bergen Point, N. J., where he introduced improved methods for the manufacture of flour of sulphur, which attracted considerable attention. Between 1858 and 1863 he was associated with Captain Raht, of the Ducktown copper-mines in Tennessee, and devised a scheme of treatment for the Ducktown copper-ores, the success of which redounded greatly to the benefit of his professional reputation. I believe that Dr. Peters, in his classical work on "Modern Copper-Smelting," quotes as authority some of Dr. Trippel's laboratory-results.

About 1864 he designed the zinc-rolling mills at Bethlehem, Pa.

Somewhat later he was engaged for a period in iron- and zinc-mining in Arkansas and Missouri. After that he confined his professional operations distinctively to the mining and reduction of silver- and copper-ores in the region west of the Rocky Mountains, and especially in Idaho, Nevada and Arizona. He erected silver-mills in Idaho, and at Ione, Nevada, he introduced, for the Knickerbocker Mining Company, the lixiviation of silver-ores. In 1872, and for several years thereafter, he was the metallurgist in charge of the Manhattan mill, at Austin, Nev.

Still later he was in charge of chemical works in the State of New York; but in 1878 he returned to Nevada as superintendent of the Danville silver-mines, in Nye county, and from 1879

to 1881 of the Morey mines in the same State. At about the latter date he was sent to Lower California, where he investigated the copper resources of the Boles district, making an elaborate report upon the subject. In 1882 he became, for a short time, general manager of copper-mines in the Santa Clara province of Cuba. His next position was that of metallurgist of the Old Dominion Copper Mining Company at Globe, Ariz., of which company he became superintendent in 1884. His discovery of the new and rich ore-bodies upon which that company and its successors have operated ever since, reversed the previous unprofitable experience of the enterprise. He introduced notable improvements in the timbering of the mines, modifying the "Comstock" or "Deidesheimer" system of square sets previously used. At the furnaces, he managed to make smelting profitable in spite of the then enormous cost of transporting coke and other supplies into the camp. As an instance of these commercial difficulties, it may be remarked that, during part of his incumbency, the fuel used (West Virginia coke) cost at Willcox, Ariz., on the Southern Pacific Railroad, $15.75 per ton of 2000 pounds, and the hauling by wagon from that station to Globe cost $30 per ton, making the cost at Globe $45.75 per ton. Before that time this cost had run as high as $50 and even $55 per ton. At present it is of course much lower than the lowest figures just given.

Resigning this position in 1888, Dr. Trippel devoted his attention for a year almost exclusively to private mining enterprises. About 1889 he superintended the development of the Arivaipa silver-lead mines, in Graham county, Ariz., and in 1890 became superintendent of the Buffalo Copper Mining Company, at Globe. The properties of this company had remained unworked for many years, by reason partly of the limited quantity, and partly of the refractory character of the ore exposed. Dr. Trippel opened up new ore-bodies, and worked them successfully.

In 1893 he became superintendent of the Phœnix (gold) Mining Company, at Cave Creek, Maricopa county, Ariz. But he resigned this position and turned his attention to an industry which, although outside of his experience, offered the attraction of pecuniary profit, coupled with the degree of personal repose which advancing age had led him to desire. I refer to

the business of horticulture, aided by irrigation, which is destined to make many parts of Arizona as productive of wealth as her mines. He established at Mesa, in that Territory, an extensive almond-orchard. At my last interview with him, but a few days before his death, he told me that the results of this venture had been, to some extent, disappointing, not because the favorable conditions did not exist, or the hope of ultimate profit was a delusion, but because, like every other business in which time is an element, it required sufficient capital to place the investor in the category of those to whom "all things come," because they can "wait." I am told also (and I am quite ready to believe) that the relative repose of a landed proprietor, for which he had yearned, did not satisfy him when he had secured it; and so, after a year or two of horticulture, he returned to active professional practice, becoming superintendent of the Rosemont Copper Company, in Pinal county, Ariz. It was during a visit to New York upon business connected with this company, that he contracted a cold, which developed into pneumonia, and caused his sudden death, November 26, 1896, at the age of seventy years.

Dr. Trippel was a member of the Board of Experts of the Bureau of Mines, established in New York in the 60's. This board comprised many of the foremost representatives of mining and metallurgical science in New York City, and the selection of Dr. Trippel by the trustees of the bureau as one of its members was a recognition of his already established technical ability. He became a member of this Institute in 1883. Of his connection with other scientific societies, at home and abroad, I have unfortunately no record, but I understand that he was a contributor to various government reports and technical periodicals. I possess a printed copy of a paper by him, which was read in 1865 before the American Philosophical Society.* It is entitled: "Reheating Furnace with Gas-Heating for Anthracite," and consists of an intelligent and critical description, accompanied with drawings, of Schinz's gas-generator, then a recent invention. The paper describes a reheating furnace with this attachment, of the form and dimensions of those built by John Fritz, at Bethlehem, where the author was then residing.

* *Proc. Am. Phil. Soc.*, Vol. x., 1865-68, p. 9.

As to accomplishments, it may be said that Dr. Trippel was master of several languages, and also that he was an ardent musician and a skilful performer on the flute. It is perhaps worthy of mention that, while a resident of Nevada, he, with a musical friend, discovered the talent of Miss Emma Wixom, organized for her, in Austin, Nev., her first concert, and persuaded her father to send her abroad for the musical education which resulted in her successful career as an operatic singer, under the stage-name of Emma Nevada.

Personally, Dr. Trippel was both genial and generous, as many beneficiaries of his assistance and advice can testify. In politics he was heartily a Democrat, frequently representing his party in county and territorial conventions. In religion he was an Episcopalian, but I venture to believe that denominational distinctions count for little beyond the pearly gates; and therefore I see no element of inconsistency in the circumstance that his mortal remains were consigned to their last repose in the Moravian Cemetery, at New Dorp, Staten Island.

Sulphur in Embreville Pig-Iron.

BY GUY R. JOHNSON, EMBREVILLE, TENN.

(Chicago Meeting, February, 1897.)

It is a common observation among those engaged in the iron-business, that the lower (*i.e.*, the less graphitic) grades of pig-iron show a rough face on the pig. As such irons are always low in silicon, the generally accepted belief is, that this rough face is due to the absence of silicon.

Certain investigations, however, made by the writer on Embreville pig-iron, and extending over nearly a year's time, seem to indicate another cause, namely, the giving-off of included gases, particularly sulphurous oxide (SO_2).

It is, of course, the tendency of such gases to rise through the iron, escaping into the air; and the more liquid the iron, and the longer it remains liquid, the greater the chance for such escape. The lower grades of iron, being always colder, tend to solidify early, and before the included gases can completely escape.

A certain proportion, however, does force its way up through the pasty metal, which it pushes aside; and the iron, being too cold to flow back, remains in the position into which the gas, during its passage, has shoved it, thus presenting, on cooling, a pitted appearance.

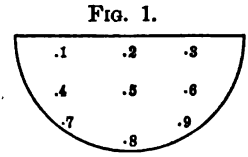
The tables given below, exhibiting the results of analysis on a number of samples of pig-metal, seem to indicate the correctness of this view. A study of these tables will show that when the iron chilled promptly, the sulphur contents were found to be much higher than otherwise.

If this theory is correct, the question arises: Is it not possible, in ordinary foundry-practice, to get rid of the sulphur by simple heating, followed by vigorous stirring of the liquid cast-iron? To test this, the writer has had several experiments made in the cupola of the foundry belonging to the Embreville estate. Owing to the small size of this furnace, it has been impossible to heat the iron to the temperature desirable. The mean of the experiments, however, showed a reduction of one-hundredth of 1 per cent. in the sulphur percentage of the iron as poured direct from the cupola, effected by pouring the same iron into a ladle and vigorously stirring it.

The writer wishes to acknowledge the services of Mr. George F. Eldridge, chief chemist of the Embreville estate, who made all the analyses required to test this theory, and whose explanatory notes are given below.

Method of Drilling.—Nine holes were drilled in the fractured end of each pig, in the relative positions shown by Fig. 1.

FIG. 1.



Relative Position of Drill-Samples.

The analyses of each drill-sample are given in the tables in the following position:

1	4	7
2	5	8
3	6	9

The tables contain also diagrams showing the form of the pig, which, as will be seen, varies considerably by reason of the wear of patterns, incomplete filling of mould, etc., and also by reason of a change made in the shape of the patterns during

TABLE I.—*No. 1 Foundry-Irons.*

	FURNACE-PIECES.				SAMPLE-PIG FROM CAST.				
			ANALYSIS.				PERCENTAGES OF SULPHUR.		
Test No.	Cast No.	Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	1-2-3	4-5-6	7-8-9
1	755	2.25	0.024	0.182	No.1 F.		0.033	0.020	0.005
							0.034	0.021	0.004
							0.030	0.020	0.005
2	203	1.37	0.021	0.237	"		0.035	0.019	0.004
							0.030	0.019	0.006
							0.032	0.020	0.007
3	819	1.25	0.013	0.254	"		0.016	0.010	0.006
							0.017	0.011	0.005
							0.018	0.011	0.005
4	811	1.94	0.019	0.262	"		0.034	0.018	0.005
							0.035	0.021	0.006
							0.030	0.021	0.005
5	217	1.09	0.023	0.181	"		0.030	0.022	0.004
							0.032	0.020	0.004
							0.033	0.019	0.004
6	840	2.10	0.016	0.144	"		0.022	0.016	0.009
							0.024	0.017	0.008
							0.024	0.017	0.008
7	193	1.05	0.023	0.229	"		0.030	0.017	0.005
							0.036	0.016	0.006
							0.032	0.017	0.006
8	220	1.95	0.023	0.182	"		0.028	0.020	0.012
							0.026	0.019	0.015
							0.026	0.018	0.011
9	679	2.00	0.011	0.175	"		0.015	0.013	0.009
							0.017	0.013	0.007
							0.019	0.012	0.010
10	208	1.37	0.021	0.237	"		0.031	0.020	0.005
							0.028	0.022	0.004
							0.032	0.019	0.005

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the period of these experiments. The various forms shown are referable to two classes, both being $3\frac{1}{2}$ inches wide at the top, and one of them $3\frac{1}{2}$ inches deep, the sides being vertical for 2 inches, and the lower part being a semicircle on a radius of $1\frac{3}{4}$ inches; while the other is 4 inches in total depth, and tapers

in width from $3\frac{1}{2}$ to $2\frac{1}{2}$ inches, the bottom being a semicircle on a radius of $1\frac{1}{4}$ inches.

Some of the diagrams in Tables V. and VI. are crossed by horizontal lines, indicating the boundary of the closeness of structure in the fracture.

Method of Analysis.—Evolution by HCl, absorption by KOH,

TABLE II.—*No. 2 Foundry-Irons.*

	FURNACE-PIECES.					SAMPLE-PIG FROM CAST.				
		ANALYSIS.					PERCENTAGES OF SULPHUR.		
Test No.	Cast No.	Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	1-2-3	4-5-6	7-8-9
11	383	1.80	0.012	0.189	No.2 F.		0.020	0.013	0.005
							0.019	0.010	0.006
							0.019	0.010	0.006
12	244	1.56	0.044	0.179	"		0.068	0.050	0.020
							0.072	0.044	0.018
							0.071	0.040	0.021
13	220	1.05	0.028	0.182	"		0.035	0.025	0.009
							0.032	0.020	0.011
							0.029	0.020	0.010
14	208	1.37	0.021	0.237	"		0.028	0.019	0.012
							0.029	0.020	0.013
							0.033	0.022	0.012
15	781	1.57	0.031	0.200	"		0.043	0.027	0.015
							0.050	0.034	0.017
							0.043	0.032	0.014
16	814	1.74	0.012	0.250	"		0.017	0.010	0.004
							0.019	0.012	0.004
							0.019	0.012	0.006
17	238	1.79	0.023	0.199	"		0.033	0.023	0.007
							0.039	0.020	0.009
							0.042	0.026	0.007
18	218	1.88	0.029	0.184	"		0.037	0.025	0.013
							0.041	0.023	0.015
							0.044	0.025	0.017

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titration with iodine. Embreville iron parts with all its sulphur by evolution-methods; but in this work about one-tenth of the residues were examined for sulphur. None was found. Of the total number of residues examined, about one-half were from drillings of the close portions of the pig.

Notes on the Work.—The work includes all pigs examined, 43 in number. Of these, 25 (Tables I., II. and III.), having the fracture of No. 1, 2 and 3 foundry, show the sulphur at the

TABLE III.—*No. 3 Foundry-Irons.*

FURNACE-PIECES.			ANALYSIS.		SAMPLE-PIG FROM CAST.		PERCENTAGES OF SULPHUR.		
Test No.	Cast No.	Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	1-2-3	4-5-6	7-8-9
19	524	1.54	0.084	0.146	No.3 F.		0.140	0.090	0.022
							0.142	0.080	0.028
							0.148	0.075	0.084
20	3 Fdy.Lot	1.86	0.111	0.152	"		0.152	0.090	0.088
							0.141	0.098	0.042
							0.145	0.110	0.029
21	525	1.61	0.081	0.200	"		0.052	0.084	0.018
							0.056	0.025	0.020
							0.048	0.028	0.016
22	364	1.36	0.012	0.347	"		0.020	0.013	0.008
							0.022	0.013	0.009
							0.019	0.013	0.006
23	782	1.29	0.021	0.289	"		0.042	0.038	0.015
							0.046	0.024	0.016
							0.040	0.022	0.012
24	821	1.88	0.011	0.229	"		0.015	0.010	0.005
							0.017	0.013	0.004
							0.019	0.013	0.004
25	523	2.28	0.071	0.167	"		0.113	0.082	0.051
							0.128	0.088	0.043
							0.100	0.078	0.045

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TABLE IV.—*Gray-Forge Irons.*

FURNACE-PIECES.			ANALYSIS.		SAMPLE-PIG FROM CAST.		PERCENTAGES OF SULPHUR.		
Test No.	Cast No.	Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	1-2-3	4-5-6	7-8-9
26	745	0.88	0.028	0.217	G.F.		0.026	0.025	0.022
							0.025	0.025	0.026
							0.025	0.024	0.026
27	312	0.66	0.078	0.176	"		0.082	0.081	0.079
							0.079	0.080	0.083
							0.077	0.083	0.084
28	278	0.55	0.109	0.211	"		0.115	0.121	0.110
							0.118	0.118	0.118
							0.120	0 122	0.120
29	313	0.92	0.140	0.160	"		0.152	0.143	0.150
							0.100	0.149	0.152
							0.155	0.150	0.145
30	306	1.22	0.016	0.243	"		0.016	0.019	0.019
							0.018	0.020	0.019
							0.017	0.018	0.020
31	50	2.80	0.133	0.334	"		0.128	0.130	0.139
							0.132	0.125	0.138
							0.132	0.132	0.140
32	54	2.39	0.108	0.249	"		0.100	0.110	0.118
							0.099	0.112	0.108
							0.098	0.107	0.100

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TABLE V.—*Irons not Uniform in Fracture.*

Test No.	Cast No.	FURNACE-PIECES. ANALYSIS. Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	SAMPLE-PIG FROM CAST. PERCENTAGES OF SULPHUR. 1-2-3	4-5-6	7-8-9
33	3 Fdy.Lot	1.86	0.111	0.152	No.3 F.		0.008	0.084	0.077
							0.005	0.093	0.074
							0.060	0.083	0.005
34	304	1.36	0.012	0.347	"		0.016	0.017	0.005
							0.015	0.017	0.004
							0.015	0.018	0.005
35	312	0.66	0.078	0.176	G.F.		0.127	0.125	0.113
							0.127	0.125	0.116
							0.127	0.124	0.115
36	668	0.47	0.032	0.101	" White center		0.025	0.042	0.022
							0.025	0.040	0.021
							0.025	0.042	0.022
37	313	0.62	0.140	0.100	" Mottled		0.165	0.137	0.166
							0.166	0.125	0.170
							0.180	0.137	0.180

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TABLE VI.—*Irons Giving Apparently Abnormal Resu'ts.*

Test No.	Cast No.	FURNACE-PIECES. ANALYSIS. Silicon	Sulphur	Phosphorus	Fracture	Shape of Pig	SAMPLE-PIG FROM CAST. PERCENTAGES OF SULPHUR. 1-2-3	4-5-6	7-8-9
38	193	1.05	0.023	0.220	No.1 X. Uniformly open		0.015	0.015	0.004
							0.016	0.016	0.004
							0.017	0.020	0.004
39	203	1.37	0.021	0.237	"		0.025	0.020	0.024
							0.024	0.019	0.024
							0.025	0.019	0.022
40	1	5.19	0.032	0.248	S.G. Uniform		0.025	0.025	0.019
							0.017	0.019	0.013
							0.027	0.019	0.016
41	539	1.97	0.027	0.167	No.2 F. Uniform		0.016	0.015	0.019
							0.016	0.020	0.019
							0.014	0.021	0.018
42	236	1.21	0.003	0.188	No.3 F. Close center		0.139	0.075	0.089
							0.145	0.080	0.040
							0.138	0.076	0.087
43	714	1.41	0.023	0.199	"		0.031	0.021	0.011
							0.035	0.023	0.011
							0.030	0.020	0.015

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bottom of the pig to be from one-fourth to one-half of what it is at the top, the percentage at the center being a mean. Of the remaining 18, the 7 contained in Table IV., having a gray-forge fracture, show uniform sulphur throughout the face, while 5 (Table V.), having a non-uniform fracture (close in parts), indicate an increase of sulphur at the closer points of the fracture, and 6 (Table VI.) give results not to be expected from the indications of the preceding tables. Change in the dimensions of the pigs apparently does not modify results. In many cases a single cast has furnished samples to more than one exhibit.

For uniform fracture, the mean of the three levels is very close to the results got from furnace test-pieces.

In lower grades differences are apparently smaller than in higher ones.

The Spitzkasten and Settling-Tank.

BY R. H. RICHARDS AND C. E. LOCKE, BOSTON, MASS.

(Chicago Meeting, February, 1897.)

IN "Sorting Before Sizing" (a paper first announced at the Pittsburgh meeting, February, 1896, but delayed in preparation and now presented at the present meeting) it is shown that if slime-tables are to do their best work on slimes below 0.5 mm. or 0.02 inch in diameter, they must be carefully sorted or classified, and fed to a series of tables suitably adjusted to the different slime-sorts. The reason is that the coarser and finer slime-sorts require respectively different adjustments of the tables. Hence, if the two are fed together upon a table which has, as nearly as may be, average adjustments, fine galena, for example, will go into the tails, or coarse quartz into the heads, or both—losses which could have been largely prevented if the slime had been carefully sorted and each sort sent to its own slime-tables.

As indicated in the paper just mentioned, the present tendency in this country is generally to do away with slime-sorting or "classification," and simply to take the overflow from the hydraulic classifier and divide it among the tables by means of

a distributing-tank. We must therefore look to the old country to see what means of slime-sorting have been approved.

The *Spitzkasten* is the only apparatus recognized by such authorities as Rittinger and Linkenbach. Reducing Rittinger's Austrian measures to metric units, we find that in 1866 he recommended for 100 liters per minute of slime the following dimensions:

	Width mm.	Length mm.
1st box,	100	1896.6
2d "	200	2845.0
3d "	400	3793.2
4th "	800	4741.5

And so on. As each new box is added, it widens in geometrical progression, and lengthens in arithmetical progression, the pointed boxes; twice the quantity of slime will require twice the width throughout, the length remaining the same.

Linkenbach (1887) recommends for 100 liters per minute of slime:

	Width mm.	Length mm.
1st box,	228	3600
2d "	342	4800
3d "	513	6000

With the same rule for increased quantity. Both authorities declare that 50° is the minimum angle that can be allowed for the sides of the box, but their diagrams mostly show as much as 58° for the minimum angle.

Rittinger, who was the inventor of the *Spitzkasten*, describes its action as a separation in a horizontal current according to the law of equal-settling particles. In the light of recent investigations we prefer to use the term "free-settling."* He says, further, that, in order to prevent slime from depositing, the sole of the feed-distributor should descend to the inlet of the box at an angle of 1° 15′, or 0.25 inch to 1 foot for the coarser sorts, and 0° 6.25′ or 0.02 inch in 1 foot for the finer sorts, and should enter at the exact level of the exit. Linkenbach favors raising the sole of the feed-distributor to 1 or 2 inches above the entrance, for the same purpose.

For the last box, Rittinger favors destroying the surface-current by an inverted dam, making thereby a settling-tank of

* See "The Cycle of the Plunger-Jig," *Trans.*, xxvi., 3.

this box. Linkenbach favors cutting up the boxes, when they are large, into two smaller sets, thereby discharging the spigot-products at a higher level, and diminishing the weight of the loaded apparatus.

Both authorities make note of the fact that the *Spitzkasten* gives only an approximation to true sorting according to the law of free-settling particles, but neither authority tells wherein, or how far, the *Spitzkasten* departs from perfect sorting.

In these directions the authors have sought to supply some missing data, and the lines which their investigation has taken are:

I.—The horizontal current; what is its form and what becomes of it?

II.—The feed-distributor; what is the effect upon the current of varying the angle of slope of the feed-sole?

III.—Can the current be so confined as to make it a true horizontal current?

All these investigations require some means of coloring the liquid composing the surface-current, of seeing it when it has been colored, and of picturing it for future study and comparison. Milk of lime, added to the water, was found to be the best coloring-matter; a tank with one side of plate-glass permitted the colored current to be seen, and photography furnished the means of preserving its form, so that it could be studied at leisure.

The tank which served for a pointed box is 47.5 inches long, 23.75 inches deep and 8 inches wide (inside measures). The outlet is 2.375 inches below the top of the tank. It has a plate-glass front, and is otherwise painted black inside. Within it are two cross-partitions, 17 inches long and 8 inches wide, sloping 58° from the horizontal. One, called the tail-partition, slopes downwards and inwards from the outlet; the other, called the head-partition, slopes downwards and inwards from the end of the feed-sole at the inlet. Both are bevelled, to give a sharp, true edge of contact. They are loaded with lead, to sink them; suspended by fine wires, to support them; and wedged in place and made practically water-tight at the sides by tacking on a strip of gunny-sack. The feed-sole, 12 inches long by 8 inches wide, is packed and held by the same means as the partitions. Water is brought by two hose-pipes

and distributed by two pipes with many holes, to give an even current.

Thus made, our *Spitzkasten* is like that of Rittinger in every respect, except that we used no spigot to discharge water below, and our sides are vertical (which, indeed, is the case also in some of Rittinger's boxes).

As representing nearly the speeds of the three boxes of Linkenbach for a width of 8 inches, three rates of current were selected, namely, 190, 126 and 85 pounds of water per minute respectively.*

In our experiment the water was weighed with bucket and spring-balance. The feed-sole was placed horizontally and on a level with the outlet.

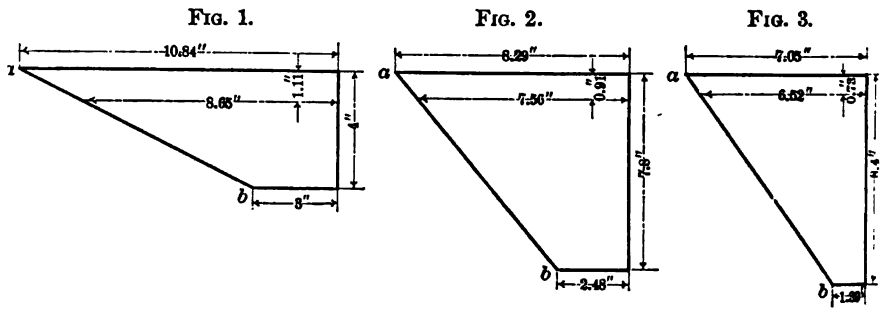


Diagram Showing Method of Calculating Overflow for Different Currents.

In each experiment the water-quantity was measured; the water-current was allowed thoroughly to establish itself; the milk of lime was added till it had just defined the main currents; and the flash-light picture was then taken.

I.—The Form of the Current.

These experiments are shown in Plates I. and II., Plate I. exhibiting the current of 190 pounds and Plate II. that of 85 pounds of water per minute. These plates show, in the first place, that the surface-current has the form of a wedge, widening (*i.e.* deepening) towards the outlet, and having, for the current of 190 pounds per minute, an apex-angle of 11°, and for 85 pounds 24.5°. The angle found for the 126-pounds current

* The exact figures should have been 196, 130.5 and 87 pounds for the width of 8 inches. The error was made by accident, but the difference is of no moment in this connection.

(not illustrated by a plate) was 16.5°. The water composing these wedges is most rapid on the surface (where the friction of the air may be disregarded), and the surface-velocity is: For 190 pounds per minute, 10.84; for 126 pounds, 8.29; and for 85 pounds, 7.05 inches per second. The velocity decreases with increasing depth, until at the bottom of the wedge it does not exceed: For 190 pounds per second, 3; for 126 pounds, 2; and for 85 pounds, 1 inch per second.

These wedges are the result of the mixing of the so-called stagnant water below with the surface-current by an action to which we shall refer as *head-suction*, or upward current on the head-partition, which widens them and slows their speed.

In the second place, it will be noticed that there is a wide under-current or eddy, returning down the slope of the tail-partition, and carrying with it a portion of the slime, which should be taken to the next box of the series, but which passes instead down the tail-partition to the bottom and then up the feed-partition to rejoin the wedge. This current shows much more plainly in Plate I. than in Plate II. In the latter, the lime had not quite reached it when the picture was taken. From this under-current the spigot obtains its water-supply, and the fine slime contained therein joins with and contaminates the slime-sort which rightly belongs to that spigot.

Thirdly, we notice in each of the plates that there is a top-layer which passes off at the outlet.

An approximate estimate was made of the two quantities: namely, that going over the outlet, and that passing down in the under-current. In Figs. 1, 2 and 3 the horizontal dimensions represent velocities; the vertical dimensions, the thickness of the two currents. The former were measured during the experiment, and the latter on the plates. Fig. 1 represents the feed-current of 190 pounds of water per minute; Fig. 2 that of 126 pounds; Fig. 3 that of 85 pounds. From these dimensions and the width of the *Spitzkasten* (8 in.) we can compute the following values:

Water per Minute. Pounds.	Overflow. Cubic Inches per Second.
190	86.6
126	57.7
85	39.6

These results were checked by weighing the overflows with scales, the result being:

Water per Minute. Pounds.	Overflow. Cubic Inches per Second.
190	87.7
126	58.0
85	39.2

We get also as the measurement of the under-current:

Water per Minute. Pounds.	Under-Current. Cubic Inches per Second.
190	134.5
126	263.4
85	230.7

These figures show an extraordinarily large quantity of slime-charged water sent down into the stagnant pool in proportion to the feed and overflow. There are, to be sure, errors in the computation. For example:

(1) The mean velocity of the top-layer for the whole length is used for the top-measure, while the point under discussion is out where the wedge forms a junction with the tail under-current, and this velocity would be less than the mean; (2) the line *a b* is assumed to be a straight line, while it may not be so. Neither of these errors is believed to affect seriously the result, which shows that the quantity of slime-charged water going away by the overflow is much smaller than that going down to revolve around in the stagnant pool. It is true that the upper layer is probably much more concentrated than the lower, and this fact is in favor of the apparatus. We note also that while the high speed (190 pounds) makes a narrower angle of wedge than low speed (85 pounds) it nearly nullifies this advantage by the high speed of the tail-undercurrent.

The authors are of the opinion that a flat layer which comes in at the feed and goes out at the outlet without forming a wedge with its tail-undercurrent and head-suction is the ideal condition to be attained, if possible, and that the more any particular form departs from that ideal the more unsatisfactory it is.

II.—The Slope of the Feed-Sole.

A series of trials was made to get the best slope of the feed-sole. The angles tested were 0°, 5°, 8°, 10°, 15°, 20°, 25°, 28° 30′ and 30°. Of these, 5° to 10° gave the narrowest

angle of wedge with all the different water-quantities. This conforms with Rittinger's and Linkenbach's requirement, namely, that fine slimes must be prevented from settling on the feed-sole.

Plate III. represents the current of 190-pound water, with 5° feed-sole slope. Angles larger than 10° tended to widen the wedge somewhat, and showed no gain to compensate for the loss of head. When 30° was reached a curious change took place. The whole current followed down the slope of the head-partition and formed a head-undercurrent, that is, a current at the feed-end, like that shown in Plate IV. Since 30° is more than half of 58°, the sole was set at 28° 30′, or about one-half of 58°; and the curious phenomenon was then observed that the feed-current was indifferent as to which of the two ways it would go, namely, as a head-undercurrent or as a horizontal surface-current. By placing a bit of iron-plate under it, it could be guided to become a surface-current, and by placing the sheet-iron over it at the proper angle, it could be deflected downwards and become a head-undercurrent. But it absolutely refused to take any intermediate position. It was either surface-current, like Plate I., or under-current, like Plate IV., and, when once directed, it would continue to flow as directed after the guide-plate had been removed.

Next a special set of variations in feed-slope was tried, comprising the following forms:

1. A catenary curve having 6-inch drop and 6-inch advance.
2. A cycloid curve, drawn by 6-inch circle, rolling on a straight line.
3. A horizontal feed-sole, depressed 1 inch below the overflow.
4. A horizontal feed-sole, raised 1 inch above the overflow, and meeting the head-partition at an obtuse angle of 122°.
5. A horizontal feed-sole, raised 1 inch above the overflow, its end being cut off square.

The catenary (No. 1) gave a higher speed than the 5°-slope feed-sole; but no gain in angle of wedge or diminution of tail-undercurrent was observed.

The cycloid (No. 2) gave the most rapid current of all; but no gain in wedge-angle or diminution of tail-undercurrent was observed.

When the horizontal feed-sole was depressed 1 inch (No. 3) it gave a wedge-angle of 25° for 190 pounds water, which was wider and more unsatisfactory than that shown in Plate II. for 86 pounds.

When the horizontal feed-sole was elevated 1 inch (No. 4) above the outlet, and the feed-sole and head-partition met at an edge obtuse angle of 122°, there resulted a head-undercurrent, shown in Plate IV., like that given by a 30°-slope, the horizontal current having disappeared altogether.

When the horizontal feed-sole was elevated 1 inch, but the end of the board was sawed off square (No. 5), a case of equilibrium was presented like that of the feed-sole sloping 28° 30′; a plate-iron guide could cause it to establish either a surface-current or a head-undercurrent, and, once established, the water would flow on in the course allotted to it, until it was disturbed.

Our conclusion is that a flat feed-sole, sloping 5°, and joining the head-partition at the level of the overflow, will give the best results.

By the experiments recorded above we are satisfied that the *Spitzkasten* is not a scientifically exact instrument. The trommel, jig, *Spitzlutte* and separator, when well constructed, and the slime-table, when properly adjusted, are all scientifically accurate instruments; that is to say, when fed with suitable material, and not overfed, they will all do perfect work. They depart from scientific accuracy only through over-feeding, or through the feeding of material that is not susceptible of perfect work, or through imperfect construction. The first two of these causes are due to the demands of commerce; the last, to ignorance.

The *Spitzkasten*, on the other hand, is scientifically imperfect. It cannot be fed with such a product, or at such a speed, or so adjusted to suit its feed, that it will do perfect work. There is always the tail-undercurrent to contaminate the product from the spigot.

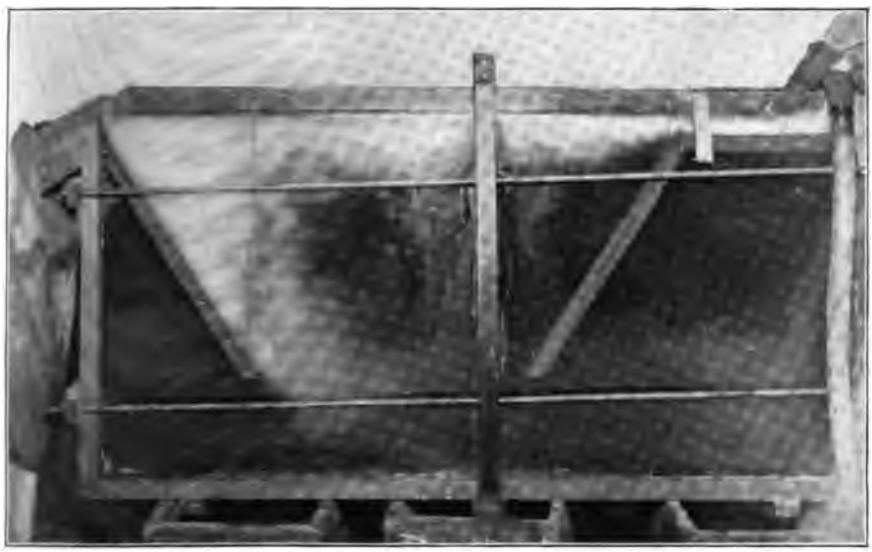
Can this defect be cured in any way? Three means have been considered, consisting in the use, respectively, of

a. A balanced hydraulic water-supply;

b. An upward current of hydraulic water

c. A perforated board

PLATE I.

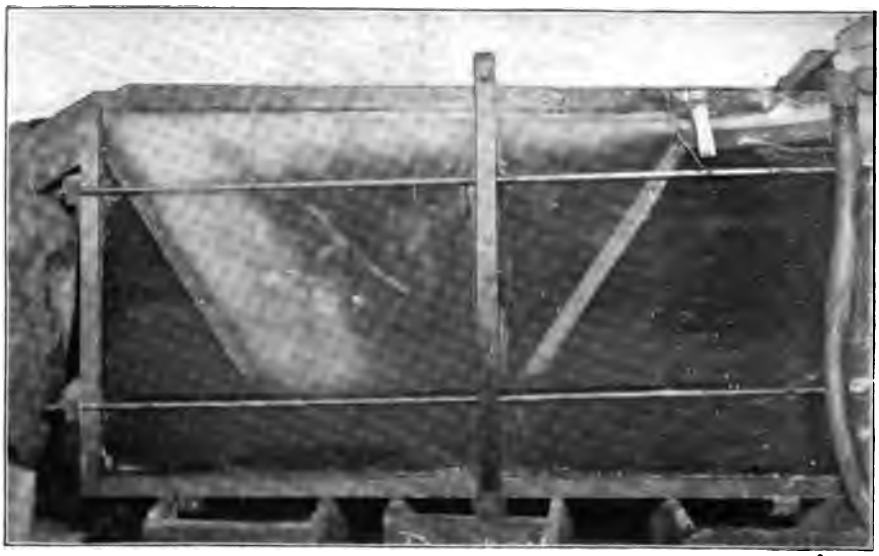


Feed Horizontal and Level with Outlet. Current, 190 Pounds per Minute.

PLATE II.

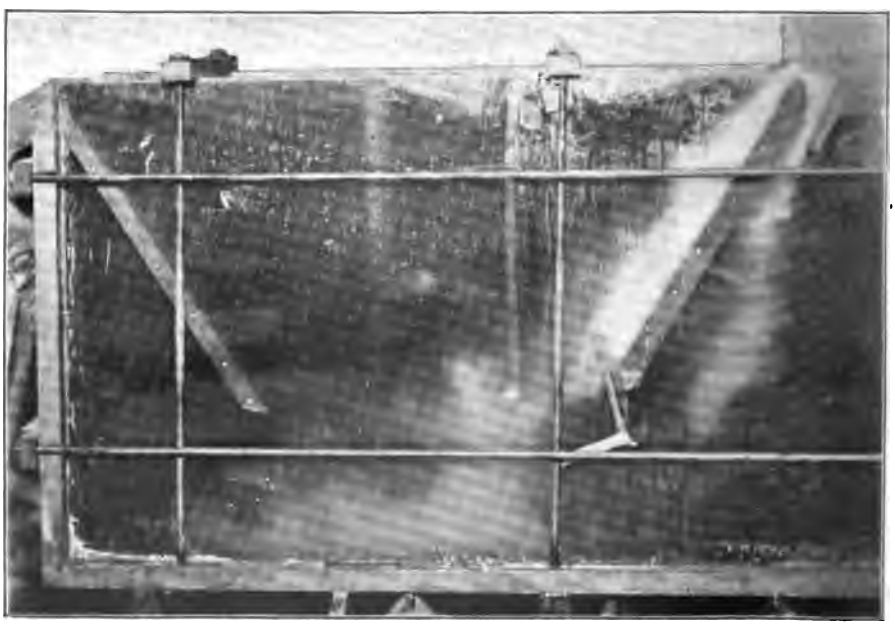
Feed Horizontal and Level with Outlet. Current, 85 Pounds Per Minute.

PLATE III.



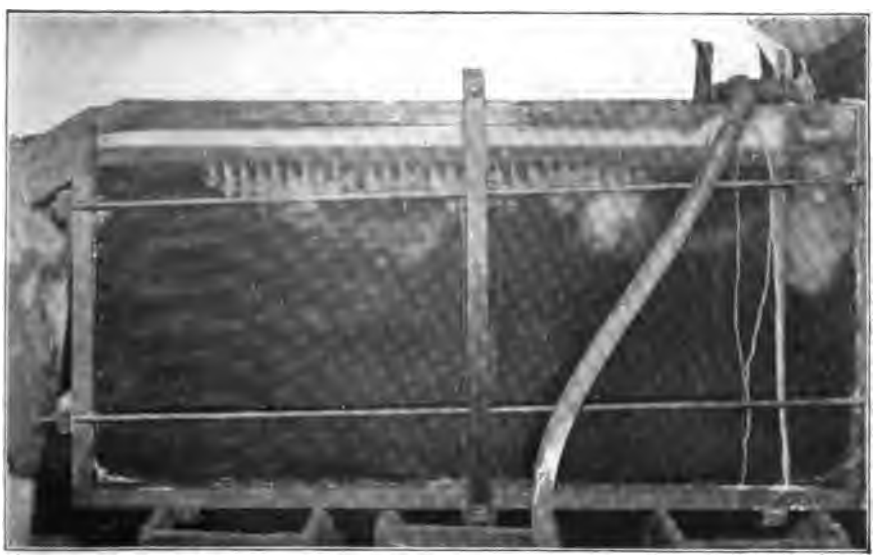
Feed Sloping 5° ; Level with Overflow. Current, 190 Pounds per Minute.

PLATE IV.



Feed Horizontal ; Elevated 1 Inch. Current, 190 Pounds Per Minute.

PLATE V.



Feed Through Level Perforated Board.

Under *a* and *b*, by balanced hydraulic water is meant an addition of hydraulic water, which will exactly supply the spigot with the right quantity of water, no more and no less. This is a difficult thing to do. A slight excess of water, that is, an upward current, will prevent the free discharge and cause a troublesome accumulation of products in the box; while a slight deficit will carry down contaminating fine slimes. This quantity, however, may be insignificant.

The method of applying this water by a vertical-tube inside the box is not approved by the authors, as it will seriously disturb the surface-current. If hydraulic balance-water is to be used, it should be put on outside with the T and plug* or T and goose-neck discharge.† The authors are of the opinion that one of these discharges, with a slight deficit of hydraulic water, is the most practical suggestion to offer.

Under *c* (perforated board), a $\frac{1}{8}$-inch board was perforated with $\frac{1}{4}$-inch holes, which were 1 inch from center to center on the cross-rows, while lengthwise the rows were 1 inch apart from center to center; but the holes of one cross-row were staggered $\frac{1}{2}$-inch with the holes of the next. This was done throughout to give the grains full opportunity to fall through the holes.

This perforated board was $47\frac{1}{2}$ inches long, of which 12 inches was blank, and served for feed-sole, while $35\frac{1}{2}$ inches was perforated, as above described. The board was placed horizontal, with its top-surface on a level with the overflow.

The apparatus was, therefore, a continuous acting *Rinne* (German) or *run* (Cornish), in which the grains suitable for deposit at any time could drop through the holes, and those not suitable could be carried forward. Plate V. illustrates a current of 190 pounds per minute. It appears very satisfactory to the eye. The mixing of fine slimes with the spigot-product is greatly reduced. The picture for 126 pounds of current was nearly as good as Plate V. But the picture, which illustrated 85 pounds current, falls away considerably, and cannot be regarded as representing good work.

This perforated board must have, at least, a balanced hydraulic water. It is quite possible that if the hydraulic is in-

* See *Gold Amalgamation*, McDermott and Duffield, Plate III., Fig. 1.

† See Rittinger's "*Spitzlutte*," in his *Aufbereitungskunde*, Fig. 18*J b*.

troduced in the stagnant-water space, with means to break up its current, a noticeable upward current could be used, which would go far towards keeping out altogether the objectionable fine slimes, and towards the yielding of a product truly sorted, according to the law of free-settling particles. Such a hydraulic should be applied as a horizontal current for the whole width of the box, and directly beneath the feed-sole.

The authors are aware that a perforated board has been used in France. They, however, did not know of its existence until after the work here recorded was done. They are aware, also, that the Kohinoor mill, in Missouri, has a *Spitzkasten* with a first box supplied with a horizontal lattice-work.

The Settling-Tanks.

The overflow from a *Spitzkasten* with balanced hydraulic water must go to a settling-tank to extract the pulp which is to feed the last table, as the slime-water has not been lessened, and may have been increased, in volume by its passage through the *Spitzkasten*.

The plates accompanying this paper throw a good deal of light upon settling-tanks. A settling-tank which is fed with a horizontal current at the surface will have a bad surface-current of wedge-shape, which will probably have an angle not narrower than 12° or wider than 25°, according to the way the current is introduced. If it is fed above the surface, it will produce a head-undercurrent, which is as bad as the surface-current. If it is fed with a plunging stream, this will pass to the bottom and flatten and form a bottom-undercurrent. All these ways of feeding are, therefore, bad.

To make the most of a settling-tank, the current of feed-slime must be brought to approximate rest as soon as possible after entering the tank, and a very slow movement must be established, which is of uniform velocity from top to bottom and from side to side. An even surface-distributor across the inlet-end, deep enough to slow the current greatly, and with a bottom steep enough to prevent the settling of slime, and this followed by two gratings made up of vertical bars 1 inch square with 1 inch spaces, the bars of the second grating staggered with those of the first, will probably prove satisfactory.

The Calorific Value of Certain Coals as Determined by the Mahler Calorimeter.

BY N. W. LORD AND F. HAAS, COLUMBUS, OHIO.

(Chicago Meeting, February, 1897.)

THIS paper gives the results of experiments conducted in the metallurgical laboratory of the Ohio State University with the objects of determining: First, the calorific powers of a number of coals in general use in Ohio; and second, how far ordinary analytical methods could be used as a basis for computing these values. Incidentally a test of the performance of the calorimeter was involved.

The chemical work has been carried on in the department for some time, so that the whole investigation has extended over three or four years. The Mahler calorimeter was purchased from L. Golaz in Paris, and is practically identical with the one described and figured by Mahler in his monograph "Contribution a l'Étude des Combustibles," from the *Bulletin de la Société d'Encouragement pour l'Industrie Nationale.* A figure and description is also given by William Kent in *The Mineral Industry* for 1892, page 97.

There were furnished with the instrument three thermometers covering a range of from 7°.5 C. to 31° C. Thermometer "A" (17°.5 to 31°) was carefully calibrated by reading the ends of a thread of mercury in sixteen positions covering the whole scale, supplementing this by readings of longer threads and deducing the errors of scale by the method published by Prof. Holman in the *Technology Quarterly* of September, 1887. The maximum error of a single reading was found to be 0°.0042 C. Thermometer "B" (13° to 25°) was compared directly with "A" over the scale common to both, and a variation was found of 0°.0145 C. for 7°.5 (or 0°.002 for 1°) of the scale. Thermometer "C" (7°.5 to 19°.0) was omitted from the comparison, as it was rarely used, being only required where measurements were made at low temperatures. From the

above figures it was concluded that for the differences of temperature measured in the calorimetric determinations of the coals in question (2°.4 to 2°.8 C.) the readings could be relied upon to within 0°.005 C., which would correspond to about fourteen units.

The unit of all the heat measurements in this paper is the quantity of heat required to raise 1 gramme of water 1° C. at laboratory temperature (15° to 25° C.). The heating powers can be converted into their equivalents in British thermal units by multiplying by nine-fifths.

The water-equivalent of the calorimeter was determined by adding to the calorimeter containing the bomb in position, and supplied with 2200 grammes of water, weighed amounts of warmer water of accurately known temperature, and determining the temperature-differences produced. Three final determinations gave 472, 464 and 465 for the value sought. The value 465 was adopted in the work as being the most reliable. This constant, calculated from the weights and specific heats of the calorimeter and bomb, was 482, but owing to the fact that a portion of the calorimeter-bucket and the bomb-valve is not submerged, it is evident that this figure is too high.

The amount of water used in the calorimeter for the determination of the heating-powers was 2400 grammes. This, added to the water-equivalent of the instrument, gives 2865 as the total water-equivalent used in the calculation.

In all cases the samples of coal were prepared as for analysis by breaking up, mixing and quartering large amounts of the coal, and finally putting the last portions through a fine sieve (at least 50-mesh). They were kept in tightly corked bottles until used. Those received from Dr. Dudley came in glass-stoppered bottles. The samples so prepared were used both for the analysis and for the calorimetric test. In the case of a few of the coals the samples were in large glass jars (having been through $\frac{1}{8}$-inch sieves), and the analysis and calorimetric determination was made on different small samples prepared from this same large one. The objection noted by Mahler to the use of a finely pulverized sample, as giving loss from coal blown away by the oxygen entering the bomb, was overcome by the method given by Hempel (*Gas-Analysis*, p. 355). The pulverized coal was compressed under a screw-press by a steel piston

in a hole bored in a block of cast-iron, forming a little briquette which was used in the combustion. No trouble whatever was experienced in getting perfectly coherent cylinders from any of the coals tested. In fact these would stand quite rough handling without breaking. The briquette of coal was accurately weighed before putting it into the tray of the calorimeter. The amount taken never varied more than 25 milligrammes from 1 gramme, and all results were calculated to 1 gramme. It will be seen from this that all the determinations of heating-power given in this paper represent the coal in question as accurately as a chemical analysis could do it, so far as sampling is concerned. This would certainly not have been the case if anything but finely powdered and carefully mixed samples had been made the basis of the work. The oxygen-pressure used in the bomb in our experiments was only 16 to 17 atmospheres. While this pressure is considerably lower than that used by Mahler, it has been conclusively shown by Hempel (*Gas-Analysis*, p. 353) to be amply sufficient for complete combustion. Furthermore, an analysis of the gas from the bomb after one of our experiments showed the entire absence of CO and the presence of over 64 per cent. of O. Berthelot gives 60 per cent. of oxygen as the amount in the residual gas necessary to insure complete combustion (*Comptes rendus*, vol. cxv., p. 202, 1892). The lower pressure has the advantage of using less oxygen, and hence emptying the cylinder more slowly. The sulphur in the coal, even at this pressure, is completely burned to sulphuric acid, and can be determined in the liquid washed out of the bomb. This statement, which is made by Mahler, is supported by the following determinations. In each case the bomb was carefully washed out with distilled water, the liquid titrated for total acidity and then acidified with hydrochloric acid and filtered, the sulphuric acid precipitated by barium chloride and weighed as barium sulphate:

Sample.	Sulphur by bomb.	Sulphur by ordinary methods.
26	2.86	3.01
16	1.57	1.60
19	2.36	2.36
21	3.51	3.65
23	3.36	3.36
18	1.89	1.95
20	1.12	1.15

These tests were not made with the special object of determining sulphur, but to show the result of an ordinary calorimetric determination. The presence of a little water in the bomb before the combustion would probably prevent the trifling loss shown here in some cases.

The method given by Mahler for the calculation of the results was followed exactly. The acid contents of the bomb were titrated with standard ammonia, using methyl-orange as an indicator, as this is not affected by the iron salts invariably present in the solution. The nitric acid found, multiplied by 230, gives the heat-units to be deducted as due to the oxidation of nitrogen. As the total acidity includes that due to the sulphuric acid, Mahler's method of computing all as nitric acid would apparently lead to incorrect results. He says the error introduced is negligible in the case of the very low-sulphur coals he worked with. In some of our coals, however, the sulphuric acid produced is largely in excess of the nitric acid, as, for example, in the case of coal No. 30, where the contents of the bomb required 9 cc. of the alkali for neutralization, of which 6.4 went to the sulphuric acid and 2.6 to the nitric acid. As the sulphur in the coal is burned to SO_2 and is included among the heat-producing elements, it would seem at first that the nitric-acid correction would be much too large in this case if the total acidity was assumed to be due to nitric acid, as was done by Mahler. But, on the other hand, the sulphur in the bomb is burned to sulphuric acid and not to SO_2, and the heat produced per unit of sulphur is greater by the excess of the heat of formation of dilute sulphuric acid over that of gaseous SO_2. Berthelot gives the heat of formation of dilute sulphuric acid, which for one gramme of sulphur would be 4388 (*Mécanique Chimique*, vol. i., p. 372). Assuming the heat developed by sulphur burned to SO_2 at 2250 units this would give for the excess in question 2138 units per unit of sulphur. This shows that a deduction must be made for the sulphuric acid as well as for the nitric acid; a calculation of the difference resulting from calling all the acidity nitric acid and from making separate allowance for the sulphuric acid and the nitric acid, showed that in the case of the highest-sulphur coal examined it would only be 26 units or less than $\frac{4}{10}$ of 1 per cent. of the heating-power of the average coal. Considering the uncertainty as to

the exact value of the heats of formation involved, it has not been deemed necessary to make the separate calculation for sulphuric acid, and the results are given as calculated by Mahler from the total acidity.

The weight of iron wire used as fuse was 17 milligrammes, and the heat-development due to this was taken at 25 units.

The probable error of a single calorimeter-determination from the mean result of a large number was computed from all the results on twenty-one samples of coal on each of which more than one determination was made. There were fifty separate results on the twenty-one samples. Computing the error by the ordinary formula gave plus or minus twenty units or about $\frac{3}{10}$ of 1 per cent. as the probable error of one determination. These results were obtained by different observers and at considerable intervals of time, and include slight possible variations in the condition of the sample as to moisture and oxidation. Duplicate results obtained at the same time by the same observer frequently gave much closer checks.

The chemical analyses were made by the ordinary methods —combustion with oxygen in a glass tube containing copper oxide and lead chromate for the ultimate analyses, while the proximate analyses were made by the methods used for all the samples analyzed in this laboratory for the Ohio Geological Survey. Through the kindness of Dr. C. B. Dudley, of Altoona, we were subsequently furnished with his results on all the coals sent to us by him. The agreement on ash and sulphur with our own figures was in all cases very close. The fixed carbon and volatile combustible determinations could not be directly compared, as his results did not state the percentage of moisture as he found it in the sample. Therefore on these items our figures were not checked directly by his and can, of course, only be compared among themselves, but as all our results on fixed carbon were by identically the same method they must be regarded as consistent with each other. In outline the treatment was as follows:

One gramme of the coal was dried at 100° to 105° C. for one hour in a crucible, the loss being called moisture. After drying, the same portion was heated three and one-half minutes over a Bunsen burner, then three and one-half minutes over a blast-lamp, and the loss was called volatile combustible. The

crucible was tightly covered and not allowed to cool during the change from burner to blast-lamp.

The samples of coal used in these experiments were as follows: Nos. 1, 2, 3, 4 and 5 were samples of Pocahontas coal, "run-of-mine," which were taken at various times from coal in use at the University boiler-house. The samples represent large amounts of coal, and were obtained by taking a shovelful from each barrow-load sent to the boilers during runs of from twelve to thirty-six hours. The coal obtained was sampled by quartering, as described before. Samples 6, 7, 8, 9 and 10 were of Hocking Valley coals taken in the same way. Nos. 11, 12, 13 and 14 were similar samples of Thacker coal. Samples 15 to 29 inclusive were sent to the laboratory through the kindness of Mr. G. L. Potter and Dr. C. B. Dudley, of the Pennsylvania R. R., and represent coal used by that company. Samples 30 to 40 inclusive were sent to the laboratory some time ago through the kindness of Mr. S. P. Bush, of the Pennsylvania R. R. Each of these samples consisted of about fifty pounds of coal taken during boiler-tests on locomotives. We were furnished with exact data as to the mines from which each coal was taken, and from the location of the mine the geological position of each coal was determined for us by Dr. Edward Orton, of the Ohio Geological Survey, and Mr. L. G. Haas, of the Pennsylvania R. R.

The results of the work are given in the following tables, in which the coals of each seam are grouped together. In addition to the analytical and calorimetric data the following figures are tabulated:

1. The calorimetric power computed from Dulong's formula in this form:

$$\text{Cal. Power} = 8080\text{C} + 34{,}462\,(\text{H} - \tfrac{1}{8}\text{O}) + 2250\text{S}.$$

C, H, O and S being the amounts of carbon, hydrogen, oxygen and sulphur in one unit of the coal.

2. The difference between this result and the bomb-determination, expressed in percentages.

3. The heating-power of each sample calculated from the average heating-power of the coal-seam to which it belongs. This average heating-power was obtained by deducting from

the average of the results of the calorimeter on all the samples of the seam in question the heat due to the combustion of the average sulphur, and dividing the result by the fraction left after taking out the ash, sulphur and moisture in one gramme of the average coal. Calling this result "H," the calorimetric value of each sample was estimated from it by multiplying "H" by the fraction left after deducting from one the ash, moisture and sulphur of the particular sample and adding to the result so obtained the heat due to the sulphur of the sample, calling the calorific power of sulphur 2250.

4. The difference between the result so obtained and the calorimeter-determination, expressed in percentages.

On examining the accompanying table of results the following points appear:

In the first place, the remarkable coincidence between the heating-powers, as calculated from Dulong's formula, and the experimental determinations. In the case of the averages of the different seams we find practical identity between the heating-power as calculated from the formula based simply on the heat developed by the combustible elements, and the results of the calorimeter. This is so much at variance with the claims of many writers that, were it not the result of so many determinations, it might pass as a mere accident. The maximum difference between the heat calculated from the elementary analysis and the heat developed in the bomb is 2 per cent. of the total calculated heat, the minimum difference 0.1 per cent. The possible error of an ultimate analysis may be placed at 0.5 per cent. on carbon and 0.2 per cent. on hydrogen, especially with coals as high in ash and sulphur as are many of the samples included in our tests. This would lead to an error of about 108 units, or nearly 1.4 per cent. on the calculated heat-value. While, of course, the probable error of the ultimate analysis is less than this, it seems certainly possible that the differences between the observed and calculated heat-values are within the limits of experiment.

Taking the differences in groups, we find as follows, including all the samples with ultimate analysis:

Table of Results.

Upper Freeport Coal, Ohio and Pennsylvania.

Sample Number.	Location of Mine.	Carbon.	Hydrogen.	Oxygen.	Nitrogen.	Sulphur.	Ash.	Moisture.	Volatile Combustible.	Fixed Carbon.	Calorimeter Result.	Calorific Power Calculated from Ultimate.	Difference.	Per cent.	Calorific Power Calculated from "H."	Difference.	Per cent.
21	East Palestine, O.	70.58	4.88	7.76	1.24	3.65	11.89	0.82	34.98	52.65	7109	7132	—	.3	7105		0.0
22	" "	73.23	5.15	8.82	1.47	1.75	9.58	1.65	37.45	51.32	7330	7352	—	.3	7346	—	.2
23	Waterford, O.	74.39	5.15	7.80	1.40	3.44	7.82	1.55	37.29	53.31	7459	7529	—	.9	7398	+	.8
24	Yellow Creek, O.	73.15	4.98	7.41	1.40	3.89	9.17	1.23	38.72	50.88	7464	7393	+1.0		7285	+2.6	
25	Steubenville, O.	74.73	5.26	8.06	1.44	2.85	7.66	1.47	39.23	51.54	7504	7567	—	.8	7455	+	.7
26	Cambridge, O.	70.61	5.19	10.33	1.44	3.01	9.42	2.43	37.79	50.36	7088	7116	—	.4	7217	—1.8	
30	Steubenville, O.	71.40	4.62	10.64	1.20	3.00	9.10	2.40	39.20	49.30	7113	6970	+2.0		7230	—1.6	
31	Salineville, O.	72.62	5.13	9.92	1.23	3.00	8.10	2.80	36.30	52.80	7271	7276	—	.1	7297	—	.4
36	Palestine, O.	71.29	5.00	9.28	1.34	2.64	10.45	2.15	36.70	50.70	7277	7136	+2.0		7176	+1.4	
37	New Galilee, Pa.	73.57	5.20	8.94	1.35	2.24	8.70	2.30	36.70	52.30	7267	7401	—1.8		7335	—	.9
40	Palestine, O.	73.64	5.06	9.47	1.24	2.34	8.25	2.45	36.60	52.70	7344	7340	+	.1	7355	—	.2
	Average	72.65	5.06	8.95	1.34	2.89	9.10	1.98	37.35	51.63	7298	7292					

$$100 \times \frac{\text{Average Calorimeter Result} - \text{Average Sulphur} \times 22.5}{100 - \text{Average (Ash} + \text{Sulphur} + \text{Moisture)}} = \text{"H"} = \frac{7293 - 65}{.8608} = 8398$$

Pittsburgh Coal, Pennsylvania.

15	Carnegie, Allegheny Co., Pa.	77.20	5.26	8.51	1.68	1.42	5.93	1.45	36.42	56.20	7691	7717	—	.3	7738	—	.6
16	Turtle Creek, " "	76.56	5.22	7.00	1.67	1.60	7.95	1.08	34.88	56.59	7630	7719	—1.2		7574	+	.7
27	Carnegie, " "	76.57	5.13	8.82	1.64	1.76	6.08	1.07	37.79	55.06	7765	7614	+2.0		7724	+	.5
28	" " "	73.50	5.19	8.08	1.44	2.54	9.25	1.08	37.67	52.00	7396	7436	—	.5	7407	—	.1
29	Creedmore, " "	74.45	5.27	8.02	1.60	1.80	8.86	1.09	38.91	51.14	7496	7528	—	.4	7484	+	.2
34	N. Mansfield, " "	73.91	5.15	8.89	1.28	1.77	9.05	2.10	36.20	52.65	7354	7404	—	.7	7386	—	.4
38	Turtle Creek, " "	74.48	5.05	8.39	1.37	1.66	9.05	1.75	36.20	53.00	7394	7433	—	.5	7421	—	.4
	Average	75.24	5.18	8.24	1.51	1.79	8.02	1.37	36.80	53.81	7582	7550					

$$\text{"H"} = \frac{7532 - 40}{.8882} = 8435$$

Middle Kittanning (Darlington Coal), Pennsylvania.

17	Hoytdale, Lawrence Co., Pa.	77.83	5.22	9.38	1.65	1.57	4.35	1.60	36.40	57.65	7785	7719	+	.9	7774	+	.1
18	Beaver Creek, " "	74.60	5.06	8.23	1.40	1.96	8.75	1.50	34.33	55.42	7360	7460	—1.3		7390	—	.4
19	Wampum, " "	77.93	5.17	7.95	1.65	2.35	4.95	0.75	38.53	55.77	7825	7787	+	.5	7747	+1.0	
20	Near Wampum, " "	76.81	5.22	8.52	1.62	1.18	6.65	0.70	36.80	55.85	7638	7663	—	.3	7681	—	.5
*32	Hoytdale, " "	72.78	4.93	10.57	1.34	1.68	8.70	2.70	35.10	53.50	7245	7173	+1.0		7311	—	.9
*33	Wampum, " "	72.82	5.25	8.55	1.33	3.25	8.80	2.85	37.50	50.85	7304	7395	—1.2		7194	+1.5	
35	Clinton, " "	73.57	5.14	10.14	1.24	1.86	8.05	2.55	35.60	53.80	7310	7320	—	.3	7366	—	.9
	Average	75.19	5.14	9.05	1.46	1.98	7.18	1.81	36.32	54.69	7494	7502					

* 32 and 33 are from the same mines as 17 and 19, but taken some years earlier.

$$\text{"H"} = \frac{7494 - 44}{8903} = 8368$$

Table of Results—Continued.

Middle Kittanning (Hocking Valley Coal), Ohio.

Sample Number.	Location of Mine.	Carbon.	Hydrogen.	Oxygen.	Nitrogen.	Sulphur.	Ash.	Moisture.	Volatile Combustible.	Fixed Carbon.	Calorimeter Result.	Calorific Power Calculated from Ultimate.	Difference.	Per cent.	Calorific Power Calculated from "H."	Difference.	Per cent.
6	Hocking Lump	69.42	5.35	16.27	1.46	1.67	5.83	6.72	37.13	50.32	6882	6790	+1.3		6836	+ .7	
7	" " 2d sample					1.63	10.10	6.45	36.60	46.85	6603				6520	+1.3	
8	Run of Mine	66.50	5.16	15.57	1.43	1.67	9.67	6.65	34.14	49.54	6496	6520	— .4		6537	— .6	
9	" " " 2d sample					1.50	10.53	6.34	35.18	47.95	6482				6503	— .3	
10	" Lump, 3d sample	68.18	5.36	15.09	1.44	1.43	8.50	6.40	36.05	49.05	6610	6740	—1.9		6662	— .8	
	Average					1.58	8.93	6.51	35.82	48.74	6612						
	Average 6–8–10	68.03	5.29	15.64	1.44	1.59	8.00	6.59	35.77	49.64	6663	6683					

$$\text{"H"} = \frac{6612 - 36}{.8298} = 7925$$

Thacker Coal, West Virginia.

11	Run of Mine	78.90	5.14	6.88	1.42	1.16	6.50	1.40	35.00	57.10	7768	7876	—1.3		7811	— .5	
12	" " 2d lot					1.18	7.50	1.60	34.75	56.15	7738				7707	+ .4	
13	Nut Coal					1.81	7.30	1.18	36.07	55.45	7711				7723	— .1	
14	" " 2d lot	78.40	5.19	7.56	1.40	1.40	6.05	1.35	36.35	56.25	7867	7831	+ .4		7840	+ .3	
	Average					1.39	6.84	1.38	35.54	56.24	7771				7770		
	Average 11–14	78.65	5.17	7.22	1.41	1.28	6.27	1.38	35.68	56.67	7817	7853					

$$\text{"H"} = \frac{7771 - 31}{.9039} = 8561$$

Pocahontas Coal.

1	"Run of Mine"	83.75	4.22	3.36	.85	.57	7.25	.80	18.30	73.65	8062	8089	— .3		8049	+ .2	
2	" "					.60	8.60	.75	17.05	73.60	7915				7942	— .3	
3	" "	85.46	4.25	3.24	.85	.57	5.63	.63	18.62	75.12	8185	8246	— .6		8206	— .3	
4	" "					.63	6.99	.61	17.92	74.48	8080				8083	.0	
5	" "	85.40	4.39	3.94	.85	.62	4.80	.85	18.60	75.75	8281	8258	+ .3		8256	+ .3	
	Average					.60	6.65	.73	18.10	74.52	8105				8107		
	Average 1–3–5	84.87	4.29	3.51	.85	.59	5.89	.76	18.51	74.84	8176	8198					

$$\text{"H"} = \frac{8105 - 13}{.9202} = 8794$$

Mahoning Coal.

39	Salineville, O	71.13	4.95	9.93	1.23	1.86	10.90	3.15	35.00	50.95	7032	7068	— .5				

Difference.							Number of Samples.	Per Cent. of Total Number.
Between	0.0	per cent.	and	0.5	per cent.	inclusive,	17	50
"	.5	"	"	1.0	"	"	7	21
"	1.0	"	"	1.5	"	"	6	18
"	1.0	"	"	2.0	"	"	4	11
							34	100

The order of distribution of differences is sufficiently near that of the "probability curve" to lead to the conclusion that they were determined by chance and not by a law, though there is a decided preponderance of negative errors, especially among those of small amount. The close agreement shown between these two sets of values certainly indicates that the heat-values of the coals in question, if calculated from the ultimate analysis, might be expected to lie within 2 per cent. of the calorimeter-value—a closer limit than is set by Mr. Kent in his review of the Mahler work on the French coals.

The ultimate analysis of coals is vastly more difficult to make than the calorimeter-determination; and therefore it is extremely important to know how far the ordinary proximate analysis so universally used in this country, and so rapidly made, can serve as a guide in rating the calorific powers of coals.

A relation between the fixed carbon and the calorimetric test was stated by Mr. Kent ("Heating-Power of Coal," *Min. Ind.*, 1892, p. 97); but the results of our work do not appear to correspond to his figures. Taking the Pittsburgh coal, we find the average calorific power of the samples observed to be 7532. The average ash is 8.02, the average moisture 1.37. Calculating from this the calorific power of the coal free from ash and moisture, we find it to be 8313. The average fixed carbon is 53.95, and this, calculated ash- and moisture-free, gives 59.54. Interpolating from Kent's table, this would give 8054 as the calorific power, a difference of 259 units, or 3.2 per cent. The same calculation on the average Freeport coal shows a difference of 296 units, or nearly 4 per cent.

The determination of fixed carbon is very uncertain, being much influenced by slight changes in method; therefore it is entirely possible that these differences are due to our method of analysis, giving low results as compared with that used by

the chemists furnishing his figures. We therefore investigated the results attainable by assuming the fixed carbon proportional within moderate limits to the heating-power, assuming that our results on fixed carbon were uniformly attained by the same method. In justification of this, it may not be out of place to say that the analyses were conducted with the same apparatus and in the same way that has been in use for years in the department. Applying this test to the Pittsburgh coal, we assumed that the ash- and moisture-free coal, with 59.54 per cent. fixed carbon, had the observed calorific power of 8313, and then we calculated the value of each coal from this average, assuming its calorific power proportional to the fixed carbon. The following are the results:

Sample number.	Calorific power observed.	Calculated from fixed carbon.	Difference.	
			Units.	Per cent.
15	7691	7830	139	1.7
16	7630	7900	270	3.4
27	7765	7690	75	1.0
28	7396	7260	136	1.7
29	7496	7140	356	5.0
34	7354	7350	4	.0
38	7394	7540	146	1.8

Showing that our determinations of fixed carbon could not be used for estimating the calorific power within any satisfactory limit of accuracy.

Attempts to derive a general law for all the coals examined were abandoned, and the question was taken up, how far the coal of a given deposit or seam can be regarded as of uniform quality, and its specific character determined. This has led to the interesting results given in the tables. Taking the coals of the same seam, we averaged the results of the calorimeter, and, reducing by the average ash and moisture, soon found that comparable results were obtained by regarding this value as a constant for the seam over the area examined. The sulphur, existing to a large extent as pyrites, acted as a disturbing element, and it was found better to eliminate it by calculating from the average observed calorific power the value of the coal, ash-, moisture- and sulphur-free. We assumed the value so determined to be the calorific value of the coal-vein in question. The method of calculation for the Freeport coal was as follows: Average heating-power of the samples tested, 7293; average

sulphur, 2.89 per cent.; 0.0289 times 2250, the assumed calorific value of the sulphur, gives 65 units; deducting this from the average calorific power gave 7228 as the coal-value, free from the sulphur. The average ash and moisture are 9.10 and 1.93 per cent. These added to the average sulphur give 0.1392, which subtracted from 1 leaves 0.8608; dividing 7228 by this gives 8398 as the heating-value of the Freeport coal, ash-, moisture- and sulphur-free. (This is given as "H" in the tables for the several coals.) Reversing the process and calculating the heating-power of the individual samples from this value of "H" by the formula given already, we obtain the series of values in the last column but one, and the percentage-difference between this and the true calorific value as determined by the bomb-method is given in the last column.

On comparing these results for the several kinds of coal examined, seam by seam, it would appear that the actual coal of a given seam, at least over considerable areas, may be regarded as essentially of uniform heating-value.

For the Freeport coal, the samples extend from New Galilee, Pa., to Cambridge, Ohio, a distance of perhaps 80 or 90 miles in a direct line. The samples were taken at different times and by different parties, yet, on the assumption that the average heating-value of the upper Freeport coal, ash-, moisture- and sulphur-free, is 8398 units, the individual samples in seven cases out of eleven give values within 1 per cent. of those found in the calorimeter, and in only one case differed by more than 2 per cent. In the case of the Pittsburgh coal, the correspondence is still closer. The heating-power of the actual coal of this seam is 8435; and taking the samples from several mines, the results calculated from this value in no case differ from the observed figures by as much as 1 per cent.

In the next summary, the Middle Kittanning, we give results from two widely-separated portions of what is considered to be one seam. In these two fields there has been marked change in the quality.

The Darlington coal of Lawrence county, Pennsylvania, shows a heating-power of 8368, and the estimated value of the coals from various mines agrees with the average. The No. 6 or Big Vein coal of the Hocking Valley, Ohio, is the same seam, but widely changed in character. The heating-value of this

coal, as determined by five samples, is 7925. This vein includes most of the coal mined in Perry and Hocking counties, Ohio. The number for this seam is determined from too few samples to be entitled to great weight; but each determination represents, as accurately as they could be sampled, a number of tons of coal.

The figures for the Thacker coal and Pocahontas coal are given in the tables. These samples were made at different times from large lots of the coal as supplied to the University for steam-boilers, and represent the run of the respective coals.

The results of our tests seem to indicate the interesting conclusion that the character of a coal-seam, as far as its fuel-value is concerned, is a nearly constant quantity over considerable areas. The determination of the value for seams would be of great use, as the rapid proximate analysis, or, for that matter, merely the determination of ash and moisture in low-sulphur coals, would be sufficient to grade coals of the same vein. Of course it is dangerous to argue from so few examples; but the proposition seems reasonable. At least, we hope that further work may confirm these conclusions.

Since the table was compiled the following tests have been made in the Department Laboratory, and show the application of the value "H" for estimating the heating-values of the samples and also the comparison of the results so attained with those of the calorimeter.

Sample.	Ash. Per cent.	Moisture. Per cent.	Sulphur. Per cent.	Calorific Power. Observed.	Computed.
A, ...	3.80	7.30	0.80	7024	7000
B, ...	6.60	5.90	1.38	6863	6856
C, ...	3.65	5.02	0.87	7374	7188

The samples of the above table are all Hocking coals, but were taken at different times and from different sources. The last column but one gives the heating power as determined by the calorimeter; the final column, the same value as calculated from "H." Sample C, which shows a considerable difference between the two, was a small sample of selected lump-coal, free from slate, "bone" and pyrites, and therefore not comparable with averaged samples of the same seam.

A Decimal Gauge for Wire and Sheet-Iron.

Being an Abstract of the Report of Mr. Albert Ladd Colby to the Association of American Steel-Manufacturers.

BY R. W. RAYMOND, NEW YORK CITY.

(Chicago Meeting, February, 1897.)

THIS paper is simply a summary, prepared at the request of the Council of the Institute, of the report of Mr. Albert Ladd Colby, of South Bethlehem, Pa., presented at the New York meeting (October, 1896) of the Association of American Steel-Manufacturers, on "The Advisability of the Endorsement by the Association of the United States Standard Gauge, approved by Act of Congress, March 3, 1893, to the Exclusion of the Birmingham Gauge, the Brown and Sharpe, and all Others:" a subject on which, at the Pittsburgh meeting (April, 1896) of the said Association, Mr. Colby had been appointed as a committee. The report favors the adoption, not of the United States standard gauge to the exclusion of others, but of a decimal system of gauging all materials in thousandths of an inch, thus doing away entirely with arbitrary gauge-numbers running in inverse order to the actual thickness.

The American Institute of Mining Engineers originated this movement. The report of its committee on a standard wire-gauge, presented in October, 1877,* recommended the abandonment of the system of fixed gauges for commercial use, and of the system of representing the diameters and sizes by numbers, and the adoption, in lieu thereof, of the system of expressing sizes in thousandths of an inch or fractions of a millimeter, and the use of the micrometer-gauge as a means of measuring sizes.

In 1892 the American Society of Mechanical Engineers appointed a similar committee, and corresponding action was

* *Trans.*, vi., 500. The committee consisted of Thomas Egleston, William Metcalf, and Joseph D. Weeks.

taken by the American Railway Master Mechanics' Association, while committees of co-operation were constituted by the Canadian Society of Civil Engineers, the Engineers' Club of Philadelphia, the Civil Engineers' Society of St. Paul, and the Engineers' Club of St. Louis. The joint recommendation of the committees of the Mechanical Engineers and of the Master Mechanics was cordially approved by the others, to the effect that a decimal gauge be used for thickness of metals, the number denoting the thickness to be the dimension of the notch in thousandths of an inch.

The American Railway Master Mechanics' Association brought the discussion to a successful practical conclusion. The report of its committee comprised the following recommendations:

1. The micrometer-caliper should be used for laboratory and tool-room work, and in the shop when specially desired.

2. The solid notched gauge should be used for general shop purposes.

3. The form of this gauge shall be an ellipse whose major axis is 4 inches, the minor axis 2.5 inches, and the thickness 0.1 inch, with a central hole 0.75 inch in diameter.

4. The notches in this gauge shall be as follows (in thousandths of an inch):

2	22	60	110
4	25	65	125
6	28	70	135
8	32	75	150
10	36	80	165
12	40	85	180
14	45	90	200
16	50	95	220
18	55	100	240
20			250

5. All notches to be marked as in the above list.

6. The gauge must be plainly stamped with the words "Decimal Gauge" in capital letters 0.2 inch high, and below this the words "Master Mechanics."

7. In ordering material, the term "gauge" shall NOT be used, but the thickness ordered by writing the decimal as in the above list. (For sizes over ¼-inch, the ordinary common fractions may be used.)

The Pratt & Whitney Co. of Hartford, Conn., manufacture a solid elliptical gauge of 38 notches corresponding to the above requirements. By October, 1896, no less than seventy-two railroads, comprising practically all the leading railroads of the United States and Canada, had ordered these gauges, with the purpose of using the decimal system of measurement exclusively in their purchases of sheet-metal and wire.

United States Standard Gauge.

Number of Gauge.	Approximate Thickness in Fractions of an Inch.	Approximate Thickness in Decimals of an Inch.	Approximate Thickness in Millimeters.	Weight per Square Foot in Ounces Avoirdupois.	Weight per Square Foot in Pounds Avoirdupois.	Weight per Square Foot in Kilograms.	Weight per Square Meter in Kilograms.	Weight per Square Meter in Pounds Avoirdupois.
0000000.....	1-2	.5	12.7	320	20.00	9.072	97.65	215.28
000000.....	15-32	.46875	11.90625	300	18.75	8.505	91.55	201.82
00000.....	7-16	.4375	11.1125	280	17.50	7.983	85.44	188.37
0000.....	13-32	.40625	10.31875	260	16.25	7.371	79.33	174.91
000.....	3 8	.375	9.525	240	15.	6.804	73.24	161.46
00.....	11-32	.34375	8.73125	220	13.75	6.237	67.13	148.00
0.....	5-16	.3125	7.9375	200	12.50	5.67	61.03	134.55
1.....	9-32	.28125	7.11375	180	11.25	5.103	54.93	121.00
2.....	17-64	.265625	6.746875	170	10.625	4.819	51.88	114.37
3.....	1-4	.25	6.36	160	10.	4.536	48.82	107.74
4.....	15-64	.234375	5.953125	150	9.375	4.252	45.77	100.91
5.....	7-32.	.21875	5.55625	140	8.75	3.969	42.72	94.18
6.....	13-64	.203125	5.159375	130	8 125	3.685	39.67	87.45
7.....	3-16	.1875	4.7625	120	7.5	3.402	36.62	80.72
8.....	11-64	.171875	4.365625	110	6.875	3.118	33.57	74.00
9.....	5-32	.15625	3.96875	100	6.25	2.835	30.52	67.27
10.....	9-64	.140625	3.571875	90	5.625	2.552	27.46	60.55
11.....	1-8	.125	3.175	80	5.	2.268	24.41	53.82
12.....	7-64	.109375	2.778125	70	4.375	1.984	21.36	47.09
13.....	3-32	.09375	2.38125	60	3.75	1.701	18.31	40.36
14.....	5-64	.078125	1.984375	50	3.125	1.417	15.26	33.64
15.....	9-128	.0703125	1.7859375	45	2.8125	1.276	13.73	30.27
16.....	1-16	.0625	1.5875	40	2.5	1.134	12.21	26.91
17.....	9-160	.05625	1.42875	36	2.25	1.021	10.99	24.22
18.....	1-20	.05	1.27	32	2.	.9072	9.765	21.53
19.....	7-160	.04375	1.11125	28	1.75	.7938	8.544	18.84
20.....	3-80	.0375	.9525	24	1.50	.6804	7.324	16.15
21.....	11-320	.034375	.873125	22	1.375	.6237	6.713	14.80
22.....	1-32	.03125	.793750	20	1.25	.566	6.103	13.46
23.....	9-320	.028125	.714375	18	1.125	.5103	5.493	12.11
24.....	1-40.	.025	.635	16	1.	.4536	4.882	10.76
25.....	7-320	.021875	.555625	14	.875	.3969	4.272	9.42
26.....	3-160	.01875	.47625	12	.75	.3402	3.662	8.07
27..... ..	11-640	.0171875	.4365625	11	.6875	.3119	3.357	7.40
28.....	1-64	.015625	.396875	10	.625	.2835	3.052	6.73
29.....	9-640	.0140625	.3571875	9	.5625	.2551	2.746	6.05
30.....	1-80	.0125	.3175	8	.5	.2268	2.441	5.38
31.....	7-640	.0109375	.2778125	7	.4375	.1984	2.136	4.71
32.....	13-1280	.01015625	.25796875	6½	.40625	.1843	1.983	4.37
33.....	3-320	.009375	.238125	6	.375	.1701	1.831	4.04
34.....	11-1280	.00859375	.21828125	5½	.34375	.1559	1.678	3.70
35.....	5-640	.0078125	.1984375	5	.3125	.1417	1.526	3.36
36.....	9-1280	.00703125	.17859375	4½	.28125	.1276	1.373	3.03
37.....	17-2560	.006640625	.168671875	4¼	.265625	.1205	1.297	2.87
38.....	1-160	.00625	.15875	4	.25	.1134	1.221	2.69

The United States Standard Gauge.

The Act of Congress entitled "An Act Establishing a Standard Gauge for Sheet and Plate Iron and Steel," approved March 3, 1893, prescribes a series of gauge-numbers to be used in determining duties and taxes levied by the United States on those articles. These numbers are "based on the fact that a cubic foot of iron weighs 480 pounds. . . . . A sheet of iron 1 foot square and 1 inch thick weighs 40 pounds or 640 ounces, and 1 ounce in weight should be 1-640th inch thick. The scale has been arranged so that each descriptive number repre-

sents a certain number of ounces in weight, and an equal number of 640ths of an inch in thickness; and the weights, and hence the thicknesses, have been arranged in regular series of gradations."* The foregoing table exhibits the relations of the gauge-numbers to the British and metric weights and diameters of the metal.

Arguments Apparently in Favor of the Adoption of the U. S. Standard Gauge.

1. It has been legalized by Congress, and is in use by the Custom-House.

2. It has been adopted by forty-five sheet-manufacturers as a basis of the wages of their rollers, as well as the selling of their product.

3. The simple relationship between the gauge-numbers and the weight in ounces of a square foot of the sheet or plate.

(1) The fact that the U. S. Government has made this gauge a standard, and is using it at ports of entry, carries little weight as an argument in favor of its more general endorsement or adoption by manufacturers. Practically the U. S. Government only needs to use the gauge at the three gauge-numbers 10, 20 and 25, for the rates of duty on sheets and plates change at each of these three sizes. The gauge simply establishes the *rate* of duty; the material is actually weighed to determine its dutiable weight. Hence, the question of the relationship between gauge-numbers and weight per ounce or pound per square foot of iron or steel does not come up in this rough use of the gauge.

The force of the adoption of this gauge by the Government is further lessened by statistics, which show that the gauge was used by the Government during 1895 to gauge a weight of sheets and plates equal to only 1.46 per cent. of the annual production of sheets or plates in the United States, as the following figures from the annual statistical report of the American Iron and Steel Association show.

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* Extract from the report of the H. of R. Committee on Coinage, Weights and Measures, accompanying the bill.

Iron and Steel Sheets and Plates, Excluding Tin-Plate, Nail-Plate, Skelp-Iron and Steel.

Gross Tons.	1893.	1894.	1895.
Production in the United States,	674,345	682,900	991,459
Exported,	2,440	3,056	1,235
Imported,	37,321	28,795	14,531
Importations—percentage of U. S. Production,	5.53	4.21	1.46

(2) The fact that this gauge is in general use by sheet-manufacturers as a basis of the wages of their rollers, and the sale of their product, affects this Association* but little, inasmuch as the product of but very few of our plate-mills comes into competition with the product of the sheet-mills using this gauge, and also because it is not a practice with us to vary the rate paid per ton with the thickness of the plate rolled. Those of our members who pay by the ton generally use the weight of accepted sheared plates as a basis. The Illinois Steel Company, the only member of the Association cited by Mr. John Jarrett as having adopted the U. S. S. Gauge, reports that it is used only at the Corning Sheet Mill, at Hammond. At this mill wages are based on the scale of the Amalgamated Association of Iron and Steel Workers, but the company says that this would in no way interfere with its filling orders at this mill by the Decimal Gauge, the adoption of which, it considers, would simplify matters very much.

(3) There is no practical value to this Association* in the simple relationship existing between the U. S. gauge-numbers and the weight in ounces per square foot of the sheet or plate measured: first, because when plates are ordered by weight, customers specify the weight in *pounds* per square foot, and not *ounces* (and there is no simple relationship between pounds per square foot and these gauge-numbers, as the table shows); and, second, because the simple relationship between ounces and gauge-numbers is only true when the sheet or plate is made of *iron*.

Arguments Decidedly Against the Adoption of the U. S. Standard Gauge.

1. It simply adds another to the long list of arbitrary gauges with numbers in inverse order to thickness.

* The American Association of Steel-Manufacturers.

2. It is based on the weight of a cubic foot of iron.

3. It is limited to the measuring of sheet- and plate-iron and steel. It is not even used by the Custom House in gauging iron and steel wire and rods.

4. The thicknesses run in no systematic series, and require in some cases nine decimal places to express them, or else complicated vulgar fractions having a common denominator of not less than 2560.

5. The nominal sizes are not easy to speak: ½-inch-plate is "Number seven naught," ⅜-inch-plate is "Number three naught."

6. It will perpetuate such terms in ordering by gauge-numbers as "light" or "tight," "full" or "scant," "heavy" or "easy"—conditions which it is impossible for a roller to meet satisfactorily.

Arguments Decidedly in Favor of the Adoption of the Decimal System of Gauging.

1. The American Railway Master Mechanics' Decimal Gauge will soon be the only gauge used by the railroads and railroad-supply houses in ordering sheet-metal and wire.

2. The decimal system of gauging greatly simplifies the measuring of all gauged materials. Sheets of all thicknesses, whether iron, copper, brass, zinc, tin, glass, paper, etc., as well as all kinds of wire, rods, rivets, nails, screws, etc., can be measured by ordinary micrometer-gauges in thousandths of an inch, or, if desired, by notched gauges, the notches being simply numbered by the number of thousandths of an inch that measure them.

3. The endorsement by this Association* of the decimal system of gauging will materially aid its more speedy and general adoption, and thus help to end the inconvenience, delay and confusion which have prevailed in the past, owing to the use of the various "standard" notched gauges, which do not agree, either with themselves or in comparison with each other, as to the actual thicknesses of their arbitrary gauge-numbers.

* The American Association of Steel-Manufacturers.

The Precipitation of Gold by Zinc-Thread from Dilute and Foul Cyanide-Solutions.

BY ALFRED JAMES, GLASGOW, SCOTLAND.

(Chicago Meeting, February, 1897.)

SOME months since, in the Johannesburg gold-fields of South Africa, the attention of the author was directed to certain statements to the effect that the gold-contents of foul or extremely dilute cyanide-solutions could not be effectively precipitated in practice by the usual zinc-method.

Such statements caused all the greater surprise, because it is well known that this difficulty is not encountered in other well-known districts, as, for example, in New Zealand, where solutions containing less than 0.05 per cent. of KCy are regularly reduced in practice to below 1 grain of gold per ton, while in Mexico and the United States almost equally satisfactory precipitations are not uncommon.

As a result of these representations, however, the author visited several plants at work in the Transvaal, and noted that while at some few mines the precipitation was really excellent, at others the effluent, after treatment, was much too high in bullion-contents.

An investigation into the causes of these unsatisfactory results disclosed the fact that more attention to the close packing of the zinc-shavings in the extractor-compartments tended to improve the results, even with the most dilute solutions. The function of the zinc being one of surface, it stands to reason that a compartment well filled with zinc would be more effective than a half-filled one. Moreover, the quality of the zinc was inferior. This should be in long, tough, thin, light threads; light enough to ignite readily, and tough enough not to disintegrate easily into a slime. The coarse shavings too frequently employed appear to give the reverse of the best practice; they are brittle, contain less surface, and take up more room. It is almost impossible to pack a compartment properly

with such zinc, which permits the existence of channels through which the solution may ascend without coming into effective contact with the zinc at all.

The boxes which do the best work have compartments the depth of which is greater than any other dimension.

But even under the best conditions, solutions are encountered which give trouble when run through in the ordinary way. Such solutions are those charged with alumina salts, and those resulting from the treatment of weathered concentrates containing acid iron salts.

Very thorough investigation has been carried out on these solutions; and it is found that by simply giving them longer contact with the zinc they may be treated with complete success. With solutions the alkalinity of which has been neutralized by acid salts, it is advisable to add a slight amount of soda; but even in the most difficult case an alkalinity of 0.025 per cent., or ½ pound per ton, has been found perfectly effective.

Moreover, with extremely dilute cyanide-solutions, that is, solutions containing under 0.05 per cent. of free cyanide, perfectly satisfactory precipitation may be carried out by passing the solution through the extractors at the rate of 1 ton of solution per twenty-four hours for each cubic foot of zinc-thread employed. But even when the solutions are very foul, with iron and alumina salts in solution, and containing *no free cyanide*, one is able in practice to reduce the bullion-contents to 1 *or* 2 *grains per ton* by passing the solution through the extractors at the rate of about ½ ton of solution per cubic foot per diem, the solution being in contact with the zinc for one and one-half to two hours.

This small amount of extractor-space, even with the most difficult solutions, as compared with that required for electric precipitation, deserves to be noted.

Liquors derived from the treatment of cupriferous ores and concentrates, and containing 0.2 per cent. of copper in solution, were found to be quite amenable to a treatment similar to that which sufficed for the foul liquors from weathered concentrates.

The following are notes of experiments carried out in connection with the investigation above referred to.

The solution examined was one that was returned in a cyanide-works as containing 44 grains of gold per ton, after it had

been *twice* passed through the extractors in the usual manner. An investigation showed that it contained a considerable quantity of arsenic and nearly 1 per cent. of soluble sulphates in solution, resulting from the treatment of some exceedingly acid weathered concentrates.

The solution was run through the extractors with the following modifications:

(*a*) The solution was run through a third time at the rate of 2 tons per cub. ft. of zinc per diem.

Assay before treatment, 2 dwts. 12 grs. per ton.

Assay after treatment, 1 dwt. 18 grs. per ton.

(*b*) There were added 4 lbs. of ammonium chloride per ton of solution, and it was run through at 2 tons per cub. ft. per diem.

Assay before treatment, 2 dwts. 12 grs. per ton.

Assay after treatment, 1 dwt 17 grs. per ton.

The addition of the above amount of ammonium chloride had thus no beneficial effect at the above rate of extraction.

(*c*) There were added per ton of solution 4 lbs. of burnt lime, slaked and made into a cream, and the precipitate was allowed to settle The rate of flow was not accurately determined, owing to the precipitate having reduced the rate of flow in pipe.

Assay before treatment, 2 dwts. 12 grs.

Assay after treatment, 14 grs.

(*d*) Solution (*a*) was run through again, for a *fourth* time, at 2 tons per cub. ft. per diem.

Assay before treatment, 1 dwt. 18 grs.

Assay after treatment, 5 grs.

(*e*) The solution was run through again at especially slow rate of ½ ton per cub. ft. per diem.

Assay before treatment, 1 dwt. 11 grs.

Assay after treatment, 2 grs.

This shows a great improvement, and indicates that slow running is, of itself, more practically efficacious and cheaper than the addition of chemicals to assist precipitation.

It was found, however, that after the run had been continued for some hours, the assay of the effluent rose to 12 and then to 21 grains per ton. An examination of the zinc showed that it had been thickly coated with a white incrustation which consisted mainly of ferrocyanide of zinc.

In all the above experiments there was only a trace of free cyanide or of alkali present in solution; and the formation of the incrustation pointed to the necessity of the presence of

alkali or free cyanide in solution to prevent any such reaction.

(*f*) Some of the same solution as that used for the previous experiment was now run through the extractor at the rate of about ½ ton per cub. ft. of zinc per diem, after sufficient cyanide had been added to the solution to make it show 0.08 per cent. of free KCy.

Assay before treatment, 0 dwt. 19 grs. gold per ton.
Assay after treatment, 0 dwt. 2 grs. gold per ton.
KCy before treatment, 0.08 per cent.
KCy after treatment, 0.023 per cent.

This experiment shows that the addition of free cyanide causes a satisfactory precipitation. There was no incrustation formed on the zinc. The consumption of cyanide, however, amounted to 1 pound per ton; and to avoid this, if possible, the investigation was continued, and

(*g*) A similar experiment was carried out at the same rate of flow, but adding 4 lbs. of slaked lime per ton of solution instead of cyanide, the precipitated sludge being allowed to settle.

Assay before treatment, 0 dwt. 19 grs.
Assay after treatment, 5 grs.
KCy before treatment, *nil.*
KCy after treatment, *nil.*

The zinc was washed free from incrustation in dilute solution of caustic soda prior to the commencement of this experiment, and sufficient new zinc was added to replace the consumption of previous runs. At the conclusion of the experiment there was a slight incrustation on the zinc in the first compartment only.

(*h*) Experiment as above, but with 4 lbs. of caustic soda per ton added in place of lime. No fresh zinc was added, nor was the zinc washed.

Assay of solution before treatment, 0 dwt. 19 grs.
Assay of solution after treatment, 0 dwt. 1 gr.

After the run the zinc was very black, and there was no incrustation.

(*i*) Experiment as above, but with less soda, and the rate of flow increased to ½ ton per cub. ft. per diem.

Assay of solution before treatment, 0 dwt. 19 grs.
Assay of solution after treatment, 0 dwt. 3 grs.

(*k*) In this experiment the run was continued for some days at ½ ton per cub. ft. per diem, and samples were taken every 12 hours. There was no caustic soda

added, there being present only ½ lb. per ton of solution; gold was added to make the solution up to 2 dwts. per ton.

The solution assayed before treatment, 2 dwts. 0 gr.

The solution assayed after treatment, 0 dwt. 2 grs. to 0 dwt. 5 grs.

The above experiments show the perfect practicability of precipitation from troublesome solutions by keeping the rate of flow at ⅓ to ½ ton per cubic foot per diem, with the presence of from 0.01 to 0.15 per cent. of caustic soda.

A further test was made on the precipitation of auriferous solution carrying quantities of copper, with the following results:

(*l*) Used solution from the sumps, containing 0.12 per cent. of copper in solution; was run through extractors at the rate of ½ ton per cub. ft. zinc-space per day. Half a pound of caustic soda was added per ton of solution.

Gold in solution before treatment, 2 dwts. 0 gr.

Gold in solution after treatment, 0 dwt. 0.8 gr.

Copper in solution before treatment, 0.12 per cent.

Copper in solution after treatment, 0.097 per cent.

The zinc was coated with metallic copper, but the precipitation at above rate of flow was practically perfect.

Two more prolonged trials were then undertaken, in which a large quantity of solution was run over the zinc used in (*l*), without any addition of fresh zinc, for over a week, to note the effect of the metallic copper-coating on the zinc. It was found that the coating became disintegrated in the upper compartment, and that the precipitation continued unimpaired, though, as the quantity of zinc in the extractors was gradually lessened, the effluent solutions rose in value proportionately. On replacing, however, with fresh zinc the zinc consumed, the solutions again became practically gold-free, as in (*l*).

In connection with the above precipitation-experiments, two other points call for especial remark:

1. The amount of alkali added is kept as low as possible, to avoid any great excess beyond the amount required to neutralize the acidity of the ore and for zinc-box reactions.

2. It is well known that the presence of alkali or alkaline carbonates, always present in used solutions, interferes with the silver nitrate test, making the solution appear to be stronger in available cyanide than is really the case. Chemists in charge of works should, therefore, be careful not to be

misled by this test, or lowered extractions may result. The effect of caustic potash or soda on double cyanide of zinc and potassium is *not* to regenerate all the cyanide of potassium as shown by silver-test, the test, as suggested by C. J. Ellis, being rendered unreliable by the silver throwing out the zinc from the double cyanide, and the zinc so displaced being dissolved by the KHO present, thus prolonging the titration.

$$K_2ZnCy_4 + 2AgNO_3 + 4KHO = 2KAgCy_2 + K_2ZnO_2 + 2KNO_3 + 2H_2O.$$

The Chromite-Deposits on Port au Port Bay, Newfoundland.

BY GEO. W. MAYNARD, NEW YORK CITY.

(Chicago Meeting, February, 1897.)

FOR an account of the discovery and the determination of chromite on Port au Port Bay I am indebted to Mr. Obalski, Government mining engineer for the Province of Quebec. He writes:

"In June, 1894, with the authorization of our Government, I went to Newfoundland to examine some asbestos property. Passing through Halifax, I had interviews with some gentlemen who told me that besides the asbestos there was in the same district a big deposit of iron-ore. They handed me some specimens, and, after ascertaining that they were not magnetic, and learning that they were found in serpentine, I said they were chromic iron. I visited the place on June 27, 1894, and confirmed my opinion, pointing out at the same time the importance of the deposit, as appears in my report of July 9, 1894. I do not claim any credit for the discovery of chrome-ore in Newfoundland; but, as a matter of fact, when I came in contact with the Halifax gentlemen they did not appear to know the nature or the value of the mineral until I mentioned to them that it was chromite."

Mr. Obalski's report was mainly devoted to the asbestos-deposits. That portion relating to the chromite was as follows:

"On the right side of the creek there is a show of chromic iron of good quality, which appears scattered through the serpentine in several places on the face of the cliff for 40 to 50 feet. A few boulders are found farther up at a distance of 100 feet, and also in the creek itself. From this point the deposit exists on a relatively large area, the widest exposures being from 6 to 7 feet. Chromic iron deposits in serpentine rock have no regularity, and are merely pockets. In the case of the property in question this rule is not changed, but the pockets look to be of large extent, and could be easily worked."

My own preliminary investigation of the property was made

in September, 1895, and was followed in June, 1896, by a second and more prolonged examination.

The deposits are located at Bluff-Head, Port au Port Bay, on the west coast of Newfoundland, in lat. 48°45′ N., thirty miles from Sandy Point, on Bay St. George, which is reached by steamer from Halifax and North Sidney, Cape Breton. From the latter port the steamer skirts along the north coast of Cape Breton, touching at the fishing villages, until it reaches Cape North, when it crosses Cabot strait to Port au Basque, near Cape Ray, the southwestern point of Newfoundland. From there Sandy Point is reached in about eight hours, if the weather is good.

Bay St. George is separated from Port au Port Bay by a narrow neck of land, called "The Gravels," about 1000 feet in width. A passage formerly existed at this place; but it is now silted up by the drifting sands and gravel. The distance from Sandy Point across Bay St. George is fifteen miles, and from the Gravels across Port au Port Bay to Bluff-Head an additional fifteen miles.

Bluff-Head is a bold headland of diorite, which rises vertically from the shore about 2000 feet. Between this headland and another of smaller altitude, about one and one-half miles north, the shore-line drops down to about 100 feet, the high hills receding for one-and-a-half to two miles, and forming an amphitheater, down which pours Bluff-Head creek. This stream takes its rise in some small lakes on the high table-land back of the depression between the headlands.

So far as I have been able to determine, the country-rock is diorite, traversed by frequent and broad belts of serpentine. At the head of the amphitheater the outcropping is serpentine, with occasional massive diorite boulders.

The serpentine is of the dark-green variety, and is at times almost black.

The chromite, as in all other known deposits, is confined to the serpentine rocks, and the entire area in which the mineral has been discovered so far is controlled by the Halifax Chrome Company.

My original investigation consisted in the examination of the exposures on the mountain-side, the distance from the lowest exposure to the highest being about 300 feet, measured on the surface. Seven exposures, ranging from 2 to 8 feet in width,

were originally identified and sampled, but the work of the past summer has shown that what appeared at an intermediate point to be three separate bodies were really the outcrops of one body.

The analysis of samples from the different exposures gave a range of 39 to 50 per cent. of chromic oxide. The work of the summer of 1896 was confined to the stripping of the surface at the point where the outcrop measured 8 feet in width. After a relatively small amount of work, an exposure 97 feet in length, with a vertical face of 44 feet at the top by 46 feet on the bottom of the cutting, has been made. The ore breaks out in large angular masses, the partings between the masses being thin bands of serpentine.

Blocks of serpentine act as "headers" to the masses of chromite, but are readily separated by the blow of a sledge. In the blocks of massive chromite there are no intrusions of serpentine other than the small granular particles which are pretty regularly distributed through the mass of chromite, and which will have to be removed by dressing in order to produce a marketable product of high grade.

In addition to the stripping already referred to, openings of a more limited extent have been made on outcrops higher up the mountain, all of which bid fair to develop extensive bodies of ore.

The work of the summer of 1896 was entirely preparatory, and consisted in the stripping already referred to, the construction of a wagon-road to the shore, and the erection of houses.

About 1500 tons were mined from the open cut, of which only 200 tons were shipped; and 278 tons in addition were carted to the shore. The lateness of the season prevented the shipment of a larger quantity.

Although the sorting was done by men who had had no previous experience, a cargo of 145 gross tons shipped to Philadelphia, and there sampled and analyzed by Booth, Garrett & Blair, contained:

	Per cent.
Chromic oxide,	49.90
Silica,	6.99

The ore was sent to Pittsburgh, and was used in the manufacture of chrome-brick for basic furnaces. A complete analysis of a selected lump gave the following result:

	Per cent.
Chromium oxide (Cr_2O_3),	49.23
Iron protoxide (FeO),	17.21
Magnesia (MgO),	18.66
Alumina (Al_2O_3),	7.50
Silica, (SiO_2),	6.51
Lime, vanadium, copper and manganese,	Traces
	99.11

This and succeeding analyses were made by Dr. Elwyn Waller.

In the preliminary examination six well-defined exposures, lying one above the other within a distance of 120 feet, measured on the surface, were sampled. Beginning at the exposure farthest up the mountain, the results were as follows:

Outcrop. No.	Width. Feet.	Chromium Oxide. Per cent.
1,	2.	40.28
4,	8.	50.21
5, upper	3.	45.08
5, lower		39.03
6,	4.5.	48.92
7,	3.	47.83

The marketable standard of chrome-ore is 50 per cent. chromium oxide. For every unit above 50, from 75 cents to $1.00 extra is paid; but below 50 the deduction per unit is much greater. Low silica also enhances the value.

By close sorting and cobbing an effort is made to put a high grade of ore into the market, but the result is necessarily irregular. It is manifestly important, not only to raise the grade, but also to secure regularity of product; and this result can only be secured by a regularly organized dressing-system.

The only published record of chrome-ore concentration which I have been able to find is the report, in *The Mineral Industry* of 1893, of experiments with the California ore from San Luis Obispo county. The plant consisted of a rock-breaker, a Huntington mill, crushing to 40-mesh, and Woodbury vanners for separation. The highest results ranged from 53.76 to 54.24 per cent., the crude ore containing from 43.84 to 50.44 per cent. of oxide.

While it will be possible, with close cobbing, to ship large quantities of 50 per cent. ore from Port au Port, this could only be done by wasting much ore. In order, therefore, to design a plant for utilizing the low- as well as the high-grade ore, a careful series of dressing-tests has been carried on at the Massachusetts Institute of Technology by Prof. R. A. Richards

and his corps of assistants. I cannot express too strongly my obligation to Prof. Richards for the keen interest which he has taken in this investigation.

In the hope that he will contribute a paper to the Institute giving this series of tests in detail, I will limit myself to a statement of the final results.

The net weight of ore treated was 2237 kilos, and was passed through a Blake crusher and Cornish rolls, then through a 20-mesh sieve, and thence to a hydraulic classifier, from which a finished product was obtained; the overflow going to Harz jigs and Frue vanners, and the tails finally passing onto a revolving slime-table.

The work is automatic from the time that the ore is fed into the Blake until the final products are obtained.

The total dressed product amounted to 1690.4 kilos, or 75.54 per cent. of the crude ore, and the total of tailings was 546.6 kilos, or 24.45 per cent.

The products were as follows:

	Kilos.	Per cent. of product.	Analysis: chromic oxide. Per cent.
Hydraulic separator, 1st spigot,	562.7	33.29	56.07
Jig, 1st hutch,	660.	39.01	54.18
" 2d hutch,	140.5	8.31	55.34
" 3d hutch,	17.3	1.02	54.80
Vanner-heads,	177.	10.47	55.41
Slime-table heads,	133.4	7.89	53.81
	1690.4	99.99	

The dressed products from each source being mixed in the proper proportions, the analytical result was 55.30 per cent. chromic oxide.

The analysis of all the final products was as follows:

Mark.	Chromium Oxide. Per cent.	Silica. Per cent.
Original ore,	47.94	6.38
1st spigot,	56.07	2.07
1st hutch,	54.18	
Table-tails,	19.30	
Vanner-heads,	55.41	
Mixed products,	55.30	
2d hutch,	55.34	
Table-heads,	53.81	
Jig-tails,	24.28	
Vanner-tails,	10.07	
3d hutch,	54.80	

The plant as at present designed will handle 50 tons of crude ore daily; the crushing-capacity is sufficient for 100 tons, and dressing machinery is to be added as required.

The Chicago Main Drainage Channel.

BY J. F. LEWIS, CHICAGO, ILL.

(Chicago Meeting, February, 1897.)

MUCH has been written on this great engineering work, principally from the civil engineer's stand-point. In presenting the subject to the Institute, it seems necessary to include something of geology as well as engineering. In this case, it is particularly the glacial and post-glacial periods with which we are concerned; for it has been clearly proved that the canal recently excavated has not for the first time opened the Des Plaines valley for drainage away from Lake Michigan, and that this valley, as well as the site of this city, is very closely connected with the processes and effects of the ice period. We have simply renewed a channel and a drainage direction which formerly existed. As the days of deep water-ways for commercial purposes are fast approaching, engineers may find the consideration of ancient topography and geologic changes highly useful. For what I may say on this subject I am greatly indebted to Mr. Ossian Guthrie, who has made an earnest study of glacial geology and labored especially to decipher the records of its history in this region.*

Chicago is situated in the track of one of four great glacial ice-streams that swept down from the North: one moving due South from Marquette and entering the Illinois valley at Ottawa; one through the Lake Michigan valley, entering the Illinois valley near Lemont; one from Saginaw Bay diagonally across the States of Michigan and Indiana, entering through

* Mr. Guthrie has been identified with the interests of Chicago for the last fifty years, first as a mechanical engineer of the Old Bridgeport Pumping Works, and Chief Engineer of the Hydraulic Works of the Illinois and Michigan Canal. He built the second center turn-table bridge across the Chicago river, designed and made patterns for an engine that propelled the first tug on the river, and organized and was chairman of the first committee appointed in 1885 to report on a drainage-system for Chicago. As the plans outlined by that committee have been, in the main, carried out in the construction of the canal, Mr. Guthrie may be called "the father of the Chicago Drainage Canal."

the valley of the Kankakee; and the fourth moving westward through Lake Erie to the Wabash valley and overflowing southwest into the valley of the Illinois. Each of these streams has some distinguishing feature which enables the skilled observer to trace them back from the vicinity of Alton to their source, nearly 1000 miles away, as measured on their line of movement.

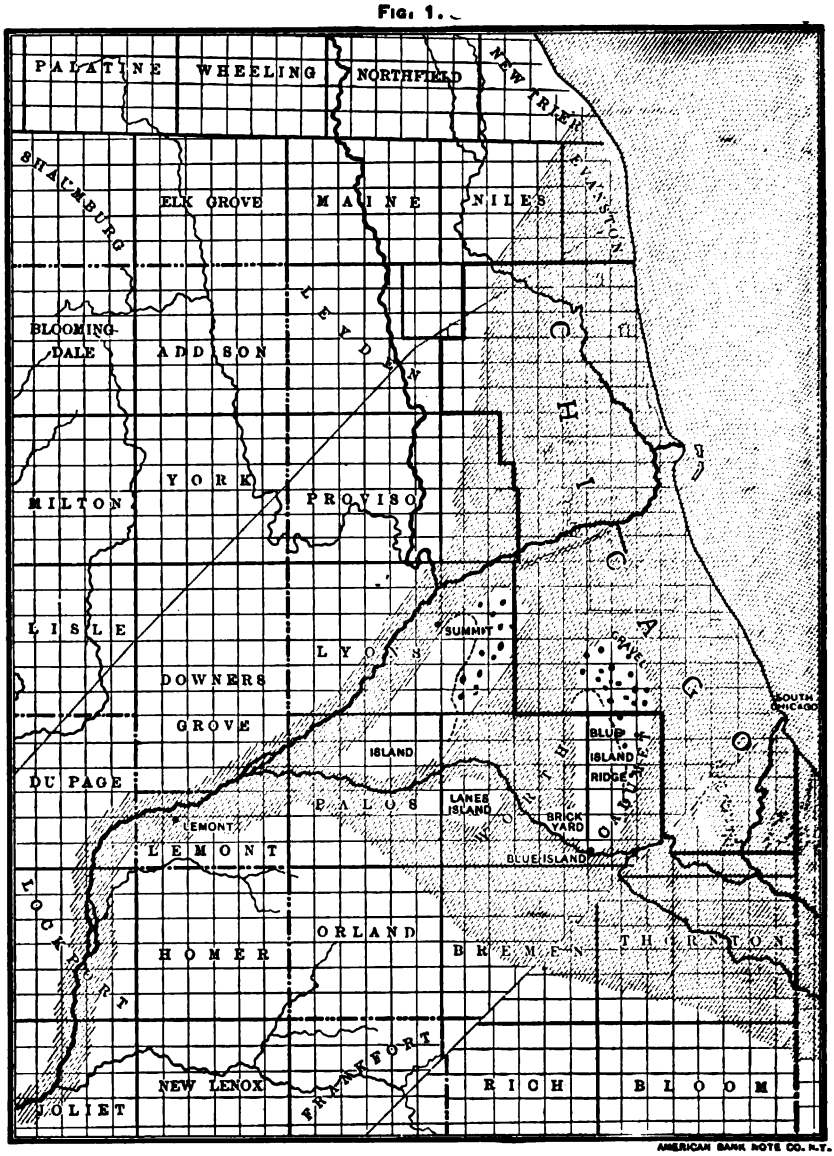
Along the paths of these streams, native copper, Huron conglomerates and granite boulders are found distributed. It would be hard to find on the globe another place of equal area where there is such a concentration of glacial evidence as within a radius of 100 miles of Chicago, or where the direction of glacial movement can be more positively determined than upon the Chicago divide between Lake Michigan and the Des Plaines valley.

This glacier (the second of those above mentioned) is supposed to have been about 125 miles wide and 700 in length, covering an area of 87,500 square miles; at Chicago, about 2500 feet deep, and at its source about 6000 feet. The average rain-fall throughout the lake-region is about 36 inches per annum, at which rate it would have required 2000 years to accumulate this quantity of ice.

Fig. 1 shows the southwest boundary of Lake Michigan during the period of glacial retreat from its south end to the Straits and while 50 feet above the present datum-line of Chicago; also, its southwestern outlet through the valley of the Des Plaines during, and only during, the same period. In this the dotted lines on Blue Island ridge show the portions of the ridge that were washed away by the waves of Lake Michigan, leaving the boulders which are represented by the large dots. The north point, marked "gravel," indicates a deposit of coarse sand and gravel carried there by the undertow of the waves. At the point marked "brick-yard," near the southern end of the island, the fine material which was carried in suspension was deposited upon the boulder-clay to the depth of 6 feet, while the sand and fine gravel was deposited along the west side and south end of the island. To the west is Lane's Island, containing a small lake about 400 feet long and 100 wide. This island shows no signs of having been affected by the waves. Northwest of it is another island or intermediate mo-

raine, about 10 miles in length. Its north end, like that of Blue Island ridge, was exposed to the action of the waves and

FIG. 1.

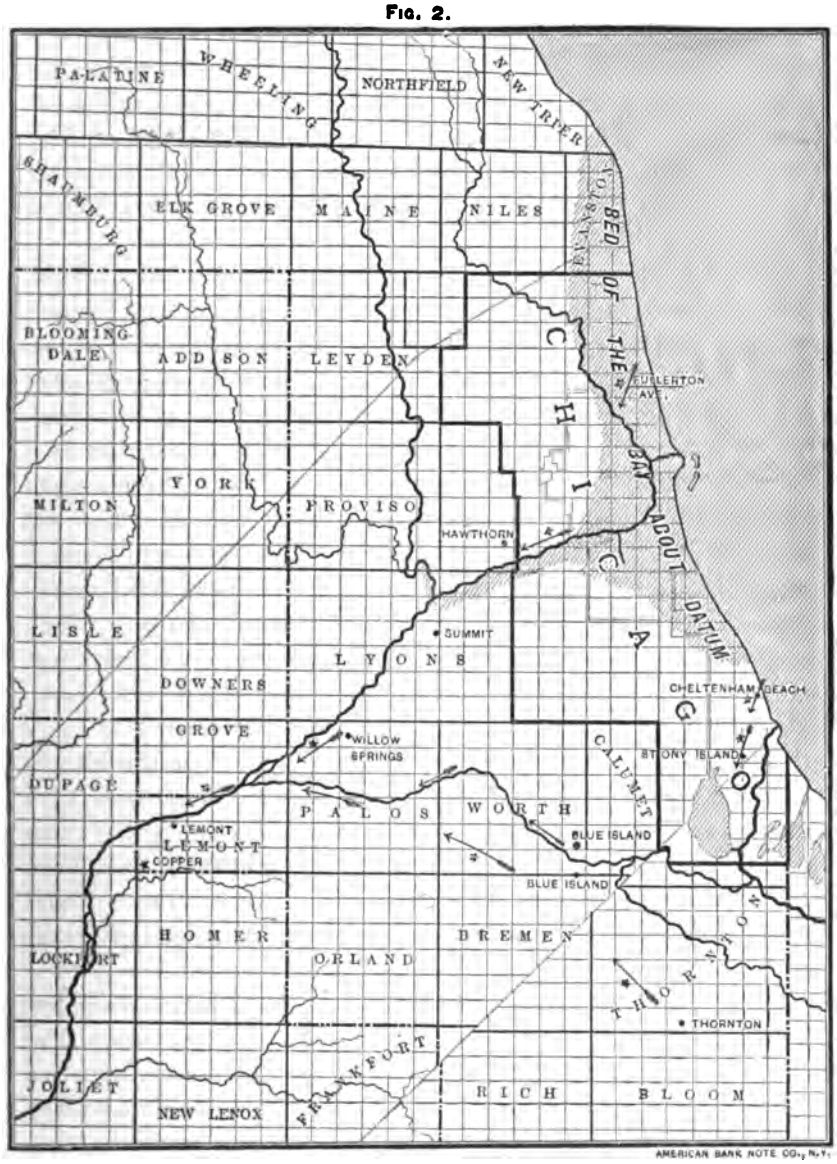


Southwest Boundary of Lake Michigan During the Period of Glacial Retreat.
(50 feet above present Chicago datum-line.)

worn away, leaving the boulders to mark its original outlines, while the gravel and sand were carried to the lee of the island and deposited.

Fig. 2 shows the boundary of Lake Michigan, or an ancient

FIG. 2.



Ancient Bay at Site of Chicago after the Disappearance of the Ice from the Straits of Mackinaw.

bay, after the ice had disappeared from the Straits of Mackinaw and the water had receded to about its present level. The bed

of this bay is almost a perfect level from north to south, but it has a slight inclination towards the lake, being a little above datum where the foundations of our present buildings rest upon it, but below datum at the lake-shore, and you will notice that the outlet from the lake down the Des Plaines valley has been closed. The point marked "copper" is where a boulder of pure copper, weighing 90 pounds, was found while stripping at one of the quarries. It was buried beneath several feet of glacial drift.

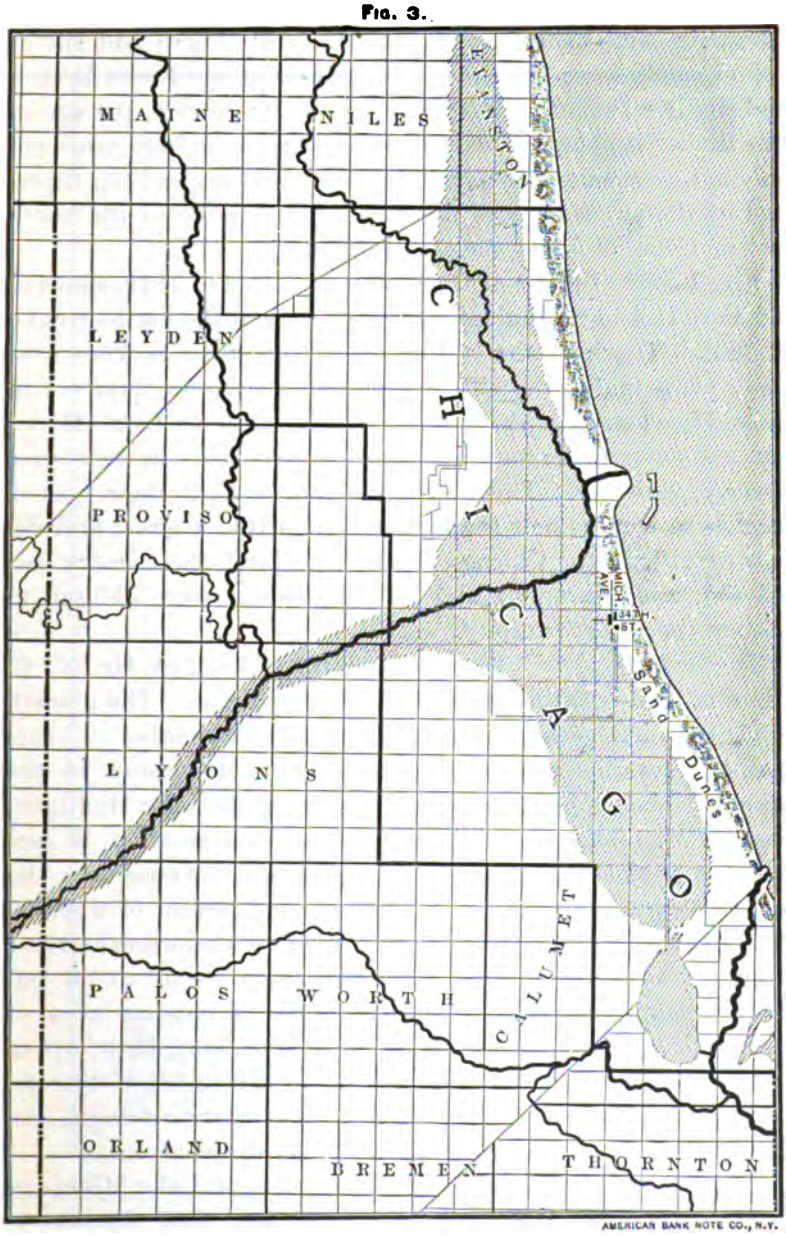
Fig. 3 shows the condition of Chicago at a still later period. The shaded portion on the map is a small inland lake formed by sand drifting across the mouth of the bay. This lake at one time stood about 11 feet above datum, and its eastern edge was at Michigan avenue and Thirty-fourth street of the present city, at which point, in excavating for the foundation of a building, many pieces of beach-worn wood were found upon the westward-sloping ancient and now buried beach. The silt from the flood-waters of the Des Plaines and the North Branch continued to fill this lake from the rear until these flood-waters broke through the sand-barrier and eroded the channel which we now call the Chicago river; and the site of Chicago was completed. After this, the silt was discharged into Lake Michigan, the ancient drainage being thus reversed.

While these great changes were taking place on the shores of the lake, an adjoining forest was evidently destroyed by a cyclone, and the trees were blown into the lake. As the lake filled up, these trees were buried, and many of them have been found in excavating for foundations and sewers. Trees found in 1871 show magnificent growth—the annual rings indicating at least 250 years' age.

Most of these trees have been found without branches, stumps or roots, and showing the effects of the cyclone. The exact date of the destruction of this forest can be only approximated. It is supposed, however, that the formation of the sand-bar upon which this timber rests began with the first north wind after the disappearance of the glacial ice at the south end of the lake,—the same wind that built 5 miles of sandy plains in the Calumet region,—and the conclusion is pretty well supported that the time occupied in forming this sand-bar represents the first epoch after the close of the glacial

period, and the one during which grew the forest of which the

FIG. 3.



Drains at Site of Chicago, Transforming the Bay of Fig. 2 into an Inland Lake.

buried timber was a part. The destruction of this forest and

its burial mark the close of the first post-glacial epoch and the beginning of the second. This takes in the filling of the inland lake with wind-drifted sand from Lake Michigan and silt or yellow sandy loam from the turbid floods of the North Branch and the Des Plaines. If this conclusion is correct, the age of this timber is approximately 6000 years; for in 8000 years not only has a considerable part of the site of Chicago been deposited by the process above described, but 5 miles of the south end of Lake Michigan have been filled up.

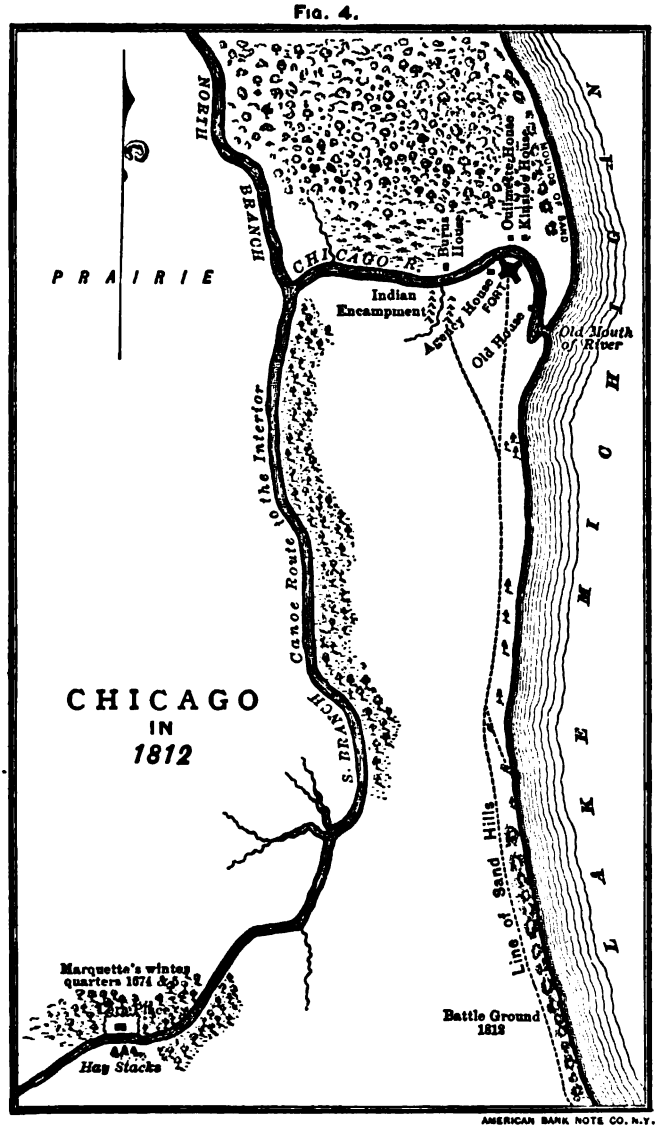
Fig. 4 shows the site of Chicago as it looked in 1812, showing old Fort Dearborn, Indian encampments, and the battle-ground of 1812. Down at the southwest corner, what is known as Lee's place marks the site of Marquette's winter quarters in 1674–75, when he with two others coasted down Lake Michigan and entered the mouth of the Chicago river, at that time covered with ice. They were obliged to haul their boat 2 leagues to what is now the intersection of Robey street and the new canal with the Chicago river. In March the country was flooded, and Marquette and his companions were obliged to take to the trees for safety.

The first house in Chicago was built in 1832 by Mr. N. C. Phillips, who is yet living and resides in Texas. The growth of Chicago has been phenomenal; and for a number of years past the question of pure water has been uppermost in the minds of the people. No doubt the water of Lake Michigan, could it be taken from beyond sewage-contamination, is perfectly pure and healthy to drink; but when one considers the fact that on a lake-front of 32 miles a population of 270,000 drains to the lake at all times, and 1,493,000 whenever the river from any cause discharges lakeward, there seems to be sufficient reason for action. The city-limits include an area of 181 square miles, or 116,320 acres. There were built, up to January 1, 1892, 4,690,338 feet or 888.321 miles of sewers, since 1855. The great question has been what to do with the sewage, and many committees have made it a study.

Fig. 5 shows the fluctuations in the level of Lake Michigan for more than half a century. At the present time the lake is steadily rising, after a period of continued depression. It is interesting to note that the levels of the other great lakes do not vary coincidently with that of Lake Michigan. Thus it is said

that at the low period of 1896 in the latter, Lake Superior showed the highest stage of water recorded in its history.

FIG. 4.



Site of Chicago after the Lake had been Filled and the Chicago River Formed.

On September 12, 1885, a supplementary committee, of which Mr. Ossian Guthrie and Mr. L. E. Cooley were members, was appointed to make a report to the executive committee of the

Citizens' Association. This report was hastened by the fall of 6¼ inches of rain, September 2d, which caused a tremendous flood to pour from the Des Plaines river, down through the Ogden ditch and the Chicago river, into the lake.

The report of this committee suggested the idea which developed into the law of 1889, creating the sanitary district and providing for the drainage channel.

In 1886 and 1887, the legislature passed bills for providing an adequate system of drainage for the city. Then came the organization of the sanitary district, October 14, 1889. The boundaries were fixed by a commission, and on November 5th were submitted to the people. The vote was: 70,958 for; 242 against. The sanitary district of Chicago is 18 miles long from north to south, and from 9½ to 15 miles wide.

Then came the formation of the Chicago Drainage Board, which, after a careful and thorough survey of different routes, finally decided upon the position of the present channel (Fig. 6). Real progress dates from December 8, 1891, upon the reorganization of the board.

In order to carry out the plan as proposed by the engineering department, it was found necessary to divert the Des Plaines river, which ran in a zigzag line through the valley. This has been done by cutting a channel 200 feet wide and 4 feet deep. The preliminary engineering work required a number of years; and it was not until June 8, 1892, that bids were opened, and work on the canal was commenced in September of that year.

The total length of the Chicago Drainage Canal, 28.05 miles, is divided into 29 sections: 14 of them practically all in dirt or glacial drift; 6 in glacial drift and rock; and 9 all in rock. There are 6 sections which exceed a mile in length, namely: Sections O, 7,022 feet; G, 5,479 feet; A, 7,300 feet; No. 1, 6,700 feet; No. 8, 5,800; and No. 10, 6,000 feet.

The dimensions are: for 14.9 miles through the rock-cut, 160 feet wide at bottom, 162 feet wide at top; and in the glacial drift for 13.15 miles, 202 feet wide at bottom, and 290 feet at the water-line when the channel is carrying 22 feet of water (see Fig. 7). The channel at the junction of the Chicago river is 24.45 feet below datum; at the Lockport end, 30.1 feet below datum. The canal-datum level was established in 1836, and the mean level of the lake for eight months in 1847 about coincided

FIG. 5.

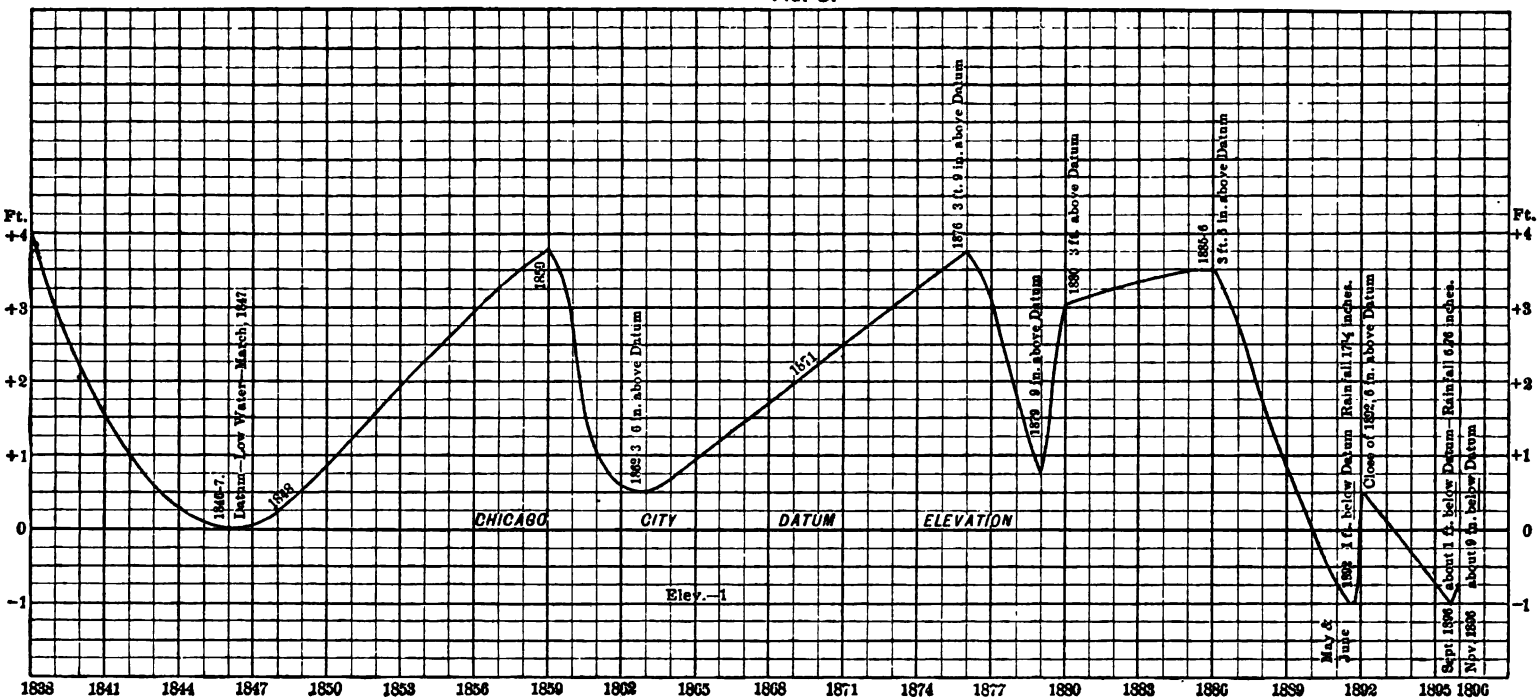


CHART SHOWING FLUCTUATIONS IN THE LEVEL OF LAKE MICHIGAN,

From 1838 to 1896, a period of 58 years. *By W. M. Christie, C.E., Chicago, 1897.*

with the canal-datum of 1836 (see Fig. 5). On the sections composed of glacial drift and rock, a retaining-wall has been built from the rock to 5 feet above Chicago datum.

The excavation of glacial drift has been 28,059,488 cubic yards; of rock, 12,343,416 cubic yards; total excavation 40,402,904 cubic yards. There are 371,969 cubic yards of retaining-wall. The average contract-price was 76.3 cents per cubic yard for rock; 28.9 cents for glacial drift; for all retaining-wall on the canal, $3.27 per cubic yard.

After the bids had been awarded the "lucky" contractors felt as if they had "drawn a white elephant," especially those who fully appreciated what they had to do in order to fulfill their contracts in the time specified, and do it at a profit. There were many special conditions on the work, to meet and to overcome which required a great deal of ingenuity and skill. Some of the sections had to be fitted up with two different plants —first for glacial drift, then for rock-excavation; and many ingenious devices for handling the material were developed. The cost of machinery and equipment was about $2,750,000; the greatest number of employés, 8500; the consumption of explosives, about 10,000,000 pounds. The maximum work excavated in one month was as follows: Glacial drift, 555,737 cubic yards; solid rock, 120,780 cubic yards; of retaining-wall there were built 45,380 cubic yards; the aggregate price was $403,464.92.

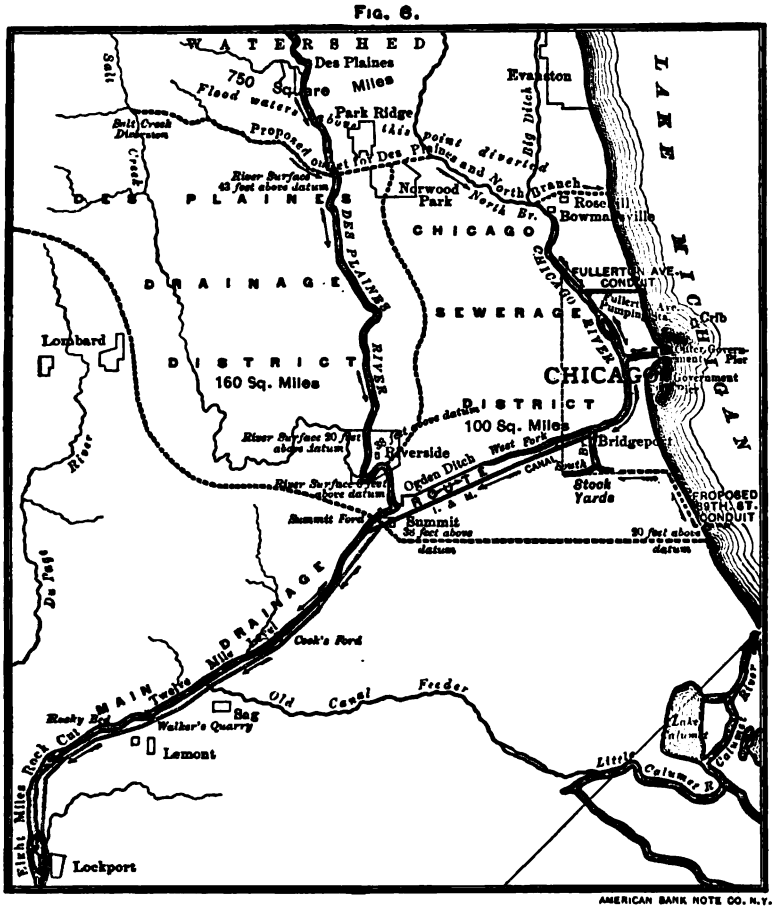
The sections of the canal are designated by numerals from Willow Springs to Lockport, and alphabetically from Willow Springs to Chicago. Commencing with Section "O" at Chicago, the work on each section will be briefly described.

Section O.—The top soil was removed by wheel scrapers loading from overhead inclined-plane platforms into cars drawn in trains by horses. A portion of the top-lift was also removed by steam-shovel, loading on to flat cars which were taken over the Illinois Central R. R. to the lake-front improvement. The remaining excavation was done with dipper dredges, delivering into barges which were towed out into Lake Michigan and dumped at places where the water is not less than 50 feet deep. The efficiency of the dredges is 600 cubic yards per day, at a cost of about 12 cents per cubic yard.

Section N.—The top-lift, to the depth of about 5 feet, was re-

moved in trains of Petler cars loaded by wheel scrapers, dumping from a raised platform through a trap. These cars were also loaded by hand-labor, pick-and-shovel method. At the present time, the remaining excavation is being made with the

FIG. 6.



Chicago Drainage District and Main Drainage Channel.

steam-shovel, incline and tipple method, as used by the Heidenreich Co. on Sections L and M.

Sections L and M.—These two sections were worked as one, the Heidenreich Co. having the contract for both. The top-soil, to the depth of from 5 to 7 feet, was removed largely by wheel scrapers, "New Era" graders and dump-cars. The remainder of the material was taken out with steam-shovels and incline-conveyors (see Fig. 8). The excavation was made lon-

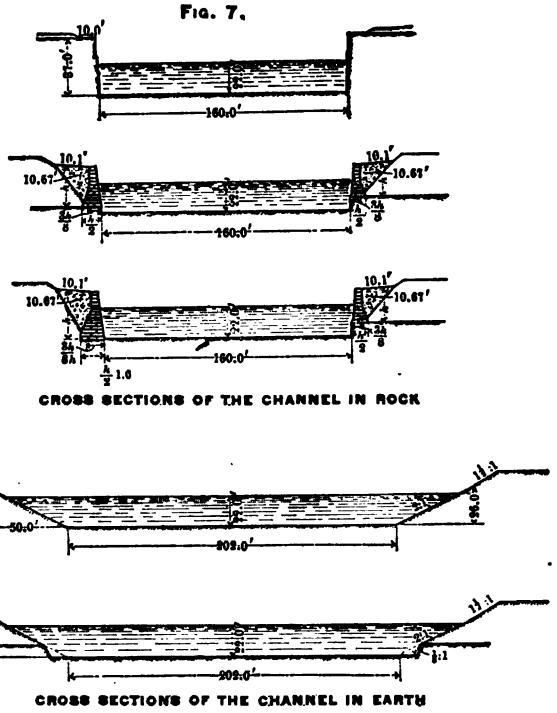
gitudinally and in two lifts of 20 feet each. The average output was 750 to 800 cubic yards per day, at a cost of about 12 cents per cubic yard.

Sections I and K.—While the main excavating-plant was being installed, the removal of the top-soil to an average depth of 5 feet from the entire surface of the two sections was effected with New Era graders and wheel scrapers. In this manner, during a period of five months, nearly 475,000 cubic yards were excavated at a cost of from 10 to 11 cents per cubic yard by the New Era graders, and from 14 to 15 cents per cubic yard by the wheel scrapers. The New Era graders are credited with about 500 cubic yards per 10-hour shift. Four truss-bridge conveyors, operated in connection with four steam-shovels, constituted the main plant for the removal of the remaining 1,800,000 cubic yards of earth or "glacial drift." This method was patented by the contractors, Christie and Lowe. The bridge-conveyor is a riveted steel truss, standing on piers which are mounted on trucks. These trucks travel on tracks laid parallel to the canal. The pier nearest the canal forms also a part of the inclined trestle-approach. This approach extends from the conveyor to the edge of the canal, a distance of 80 feet, which is the width of the berm. Two tracks extend from the steam-shovel up the slope and along the trestle and on the bridge-floor. The power-house is carried on a platform attached to the forward pier of the conveyor. Two of the conveyors had a span of 210 feet and a level floor; and two had a span of 180 feet and a floor sloping towards the cut at the rate of 1 foot in 20. The bottom-chords of the trusses are 35 feet above the natural surface of the ground, and this is the limiting height of the spoil-bank. The steam-shovel works across the channel from slope to slope, making a series of transverse cuts. The excavation was made in two stopes or lifts of about 17 feet each (see Fig. 9). The efficiency of the conveyor was about 820 cubic yards per day, at a cost of 15 cents per cubic yard.

Sections G and H.—Five methods were employed in the excavation of these two sections. A considerable portion of the surface was removed with Western wheel scrapers and with steam-shovels loading into Petler 1½-cubic-yards dump-cars, which were hauled in trains to the dump by horses.

The third means was the Bates rubber-belt conveyor. This device consisted of two rubber belts, 22 inches wide, running tandem on rollers made in the shape of a double cone. One belt extended across the cut and up onto the steel truss-bridge, which spanned the spoil-area and carried the second belt. The combined length of the two belts was 908 feet. The steam-shovel delivered the material to a brick-makers' granulator.

FIG. 7.



Sections of the Chicago Main Drainage Channel.

This cut up the clay and fed it upon the belt, which in turn transported it to the dump. The granulator was mounted on trucks parallel to the belt. The bridge carrying the belt was mounted on tracks, so that it could be easily moved forward as the excavation advanced. The average capacity of the belt was about 500 cubic yards per day, with a maximum of 1200 cubic yards.

The fourth apparatus was the Hoover and Mason steel-belt cantilever excavator (see Fig. 10). This machine spans the canal, being mounted on two cars, placed on either side of the

channel. It was first built with two cantilever-arms, so that the excavated material could be deposited on either or both sides of the canal. The excavator as thus built had a combined length of 632 feet, a height of 90 feet and a width of 20 feet. The conveyor consists of a series of steel pans, hinged together at the ends, forming practically an endless belt (see Fig. 11). The pans are made of No. 12 gauge steel, and are 4 feet from center to center of axles and 3 feet 10 inches wide. The axle carries the wheels and eye-bars which form the chain. The wheels upon which the conveyor travels are 7 inches in diameter. This belt is driven by a 1¼-inch manilla rope, which conveys the power from the engine to the end-gearing. The total length of the belt on the machine is 1000 feet, with 250 pans. Total weight of machine, 180,000 pounds. Cost of machine, about $35,000. This machine has finished its work on the canal and the total volume of material excavated by it is 241,000 cubic yards, at a cost of from 8 to 9 cents per cubic yard.

The fifth plant consisted of the Gahan and Byrne incline and tipple, which were similar in design to those used on Sections L and M by the Heidenreich Co. (see Fig. 8), but differed from the latter in that they were built of steel instead of wood, and the spoil-bank capacity was greater. These inclines hold the banner record for greatest efficiency in cubic yards of output among all the dry methods of excavation on the canal. The incline on Section H made a record in August, 1895, of 920 cubic yards per 10-hour shift, day- and night-work. One of these plants is on each of these sections.

Section F.—This, like its neighbor, Section E, has had a varied history. It is now in the hands of its third contractors, Messrs. Gahan and Byrne. Ricker, Lee & Co., the first contractors, removed nearly all of the first 8 feet of the excavation with wheel scrapers. After this, the contractors continued the excavation with a steam-shovel, locomotive and car plant, which calls for no special description. They removed 500,000 cubic yards by the above methods, when, asking for and failing to receive an increased price for the cemented boulder-clay which they encountered, they discontinued work, and on January 30, 1895, the section was re-let to Weir, McKechney & Co., who installed a bridge-conveyor plant, similar to that of

Christie and Lowe (Fig. 9), paying the latter 1 cent per cubic yard as royalty for the use of the method.

Section E.—Streeter and Kenefick, the original contractors for this section, removed some 560,000 cubic yards by the same methods as those employed on Section F by Ricker, Lee & Co.; that is to say, from 3 to 8 feet of the top-soil from the entire surface of the section was excavated with wheel scrapers, and the remainder, 160,000 cubic yards, with steam-shovel, locomotive and car plant. The contractors reached the hard material, similar to that on Section F, removed some 70,000 cubic yards of it at a cost to them of 69 cents per cubic yard, and then asked the Board to allow them greater compensation, which being denied, they discontinued work. On September 19, 1895, the section was re-let to Angus and Gindele, who at first continued the excavation by the same method as their predecessors, but in the latter part of 1895 and early winter of 1896 installed two steam-shovels, incline and tipple plants, with which they advanced the work up to December, 1896. The bid of Angus and Gindele was ½-cent per cubic yard less than that of Streeter and Kenefick, which was 27½ cents.

Section D.—New Era graders and wheel scrapers were used on this section for the removal of the top-soil. A steam-shovel, locomotive and car plant was installed for the rest of the excavation. The contractors, E. D. Smith & Co., started the steam-shovel cuts on the north side of the channel and advanced in longitudinal cuts toward the opposite side, each cut being deeper than the preceding one. The object was that when they reached the opposite side the shovels would be at grade, and would then advance toward the north side again, completing the channel. The tracks leading from the pit to the dump were located at the center of the section and extended in each direction, parallel to the canal.

Section C.—The plant for this section was similar to that just described on Section D. General William Sooy Smith operated a hydraulic dredge on the west end of the section. This dredge, instead of having a revolving cutter to loosen and mix the muck with the water, was fitted with a system of hydraulic jets.

Sections A and B.—All of Section B and the greater part of Section A extended through the bed of the Des Plaines river.

The river-bed at this point was formed by deposits of alluvium and decayed vegetation, and the depth of this material varied from 1 to 25 feet. The problem of its excavation was successfully solved by the hydraulic dredge method. The muck was mixed with the water by means of a revolving cutter at the end of the suction-pipes, where it was drawn in by powerful centrifugal pumps and forced through the discharge-pipe for a distance of sometimes 3000 feet to the discharge, or settling-basin. After the suspended material had been precipitated, the water was allowed to return by a circuitous route to the dredge-pond, and was again pumped out. From 25 to 80 per cent. of the discharge was solid matter. Two of these dredges removed 1½ million cubic yards from these two sections at a cost of about 5 cents per cubic yard. The capacity of the dredges was from 1700 to 7000 cubic yards per 10-hour shift each. After the hydraulic dredge work was finished, the sections were unwatered and plants installed for excavation of the hard material. On Section B, the east half was completed with steam-shovel, locomotive and cars, and the west half was excavated by steam-shovel, working in connection with a car-incline; the tracks extending from the pit to the dump in a continuous loop, which was humorously termed the "merry-go-round."

On Section A, two Gould conveyors were installed for the handling of the hard material. This was a method designed and patented by Mr. Charles L. Gould, C.E., of Chicago (see Fig. 12). The system has all the merits claimed for other bridge-conveyors, and others which, briefly noted, are: It grades for its own tracks and can therefore be used economically over a very rough or uneven surface, where the expense necessary to the grading of tracks for any other conveyor would greatly increase the cost of maintenance, if not be prohibitive. The spoil-bank capacity of this conveyor is equal to the height of the conveyor-deck; whereas with other types it is only equal to the height of the bottom chords of the bridge.

The capacity of the Gould conveyor is 968 cubic yards per day. This was the record made by one of these plants, in December, 1896, when in 35½ 10-hour shifts, day- and night-work, 33,431 cubic yards were excavated, at a cost of about 8 cents per cubic yard.

To prevent sloughing-in of the muck, the hydraulic dredges

removed the material from a greater width than the finished cross-section of the canal required. This necessitated the re-vetting of the sides of the canal; and this work was done ahead of the Gould conveyors by a land-dredge, shown in Fig. 13. The general dimensions of the land-dredge are as follows: Total length of car, 60 feet; total width, 18 feet. Total height of dredge above rail, 49 feet 3 inches. Maximum diameter of circle made by dipper, 152 feet. Total length of dredge with dipper-arm extended, 133 feet. Length of boom, 64 feet. Length of dipper-arm, 44 feet. Capacity of dipper, 1¼ cubic yards. The land-dredge makes a cut to level of track 90 feet wide at bottom, raises material to a maximum height of 45 feet, and deposits it 76 feet from the center of the dredge. It is mounted on 16 wheels, in 4 trucks, standing on two 30-inch gauge tracks. The total weight of the dredge is 90 tons.

Section 1.—This section was awarded to Griffiths and McDermott, June 6, 1894. They equipped it with a steam-shovel, locomotive and car-plant for the removal of the glacial drift. This was dumped on the spoil-area on each side of the channel, except 165,000 cubic yards, used to build the protection-levee between the river-diversion and the main channel on Section A. Section 1 was supposed to be the beginning of the rock-excavation; but, as a matter of fact, rock has been found on Sections F, E, D, B and A, varying from a few inches to 10 or 12 feet in depth of cutting. Section 1 has the beginning of the retaining-wall, and the transition from the walled to the unwalled channel and from 160 feet to 202 feet in the width of the bottom.

For the excavation of the rock on this section, two methods have been used; the first being the incline and tipple, differing from those described on Sections L, M, G, H and E only in the method of loading, and the car used. The incline was built of steel instead of wood. The cars were constructed of boiler-steel to meet the wear and tear of the rock, and were loaded chiefly by hand, although an air-hoist derrick on top of the working-face, for handling large masses of rock, was used. The contractors also purchased two of the three Brown cantilevers from Section 10, which are working satisfactorily (see Fig. 20).

Sections 2 *and* 4.—Sections 2, 3 and 4 were originally awarded

to McArthur Brothers, but the early stages of the excavation developed a hard material which an ordinary steam-shovel would not remove. This led the contractors to ask for a reclassification or a readjustment of the prices for the excavation of glacial drift. No satisfactory agreement was arrived at, and the sections were surrendered by mutual consent, without prejudice to either party. New bids were opened October 4, 1893, in which the prices for glacial drift ranged from 45 cents to $1 per cubic yard. On October 17, 1893, McArthur Brothers made a proposition to the Board of Trustees to do the work at the following prices per cubic yard for glacial drift: Section 2, 50 cents; Section 3, 56 cents; Section 4, 49 cents. For Sections 2 and 4, McArthur Brothers' bids made the aggregate cost of these sections less than any other bids and they were authorized to go ahead with the work. Gilman & Co. secured Section 3. The glacial drift was excavated by steam-shovels worked in connection with car-incline hoists. The steam-shovels removed or excavated the material in a series of longitudinal cuts, delivering to excavation cars (of 8 cubic yards' capacity), which were hauled by teams to the foot of the incline, where they were hoisted to the berm and thence again drawn by teams to the dump, which was mainly on the north side of the canal. After the solid rock had been sufficiently uncovered to permit its excavation to advantage, 4 Lidgerwood travelling cable-ways (Fig. 14), 2 on each section, were erected for its removal. The cable-way as a means of transporting material is not new, but its use in canal-making is new; and the construction of the Chicago drainage-canal served to develop many valuable improvements in this method, as, for instance, the portable or travelling towers and the aerial dump (Fig. 15), without either one of which the cable-way would not have been a success on the canal. The span of the cable was from 550 feet to 725 feet, with towers 93 and 73 feet high. Efficiency, 300 to 450 cubic yards per 10 hours, handled at a cost of from 28 to 30 cents per cubic yard, which included the labor of loading skips, superintendence, fuel, oil and waste, repairs, maintenance of tracks, etc., and the pay of the crew to run the cable-way. Nineteen of these cable-ways were used on the main channel and one on the river-diversion, costing $14,000 each.

All drilling on Sections 2 and 4 was done with steam.

The building of a considerable amount of the retaining-wall on the canal was sub-let. On Section 2, Winston & Co. had a sub-contract for the building of 29,135 cubic yards of retaining-wall out of a total of 36,480 cubic yards. The remaining 7345 cubic yards was built by the regular contractors, McArthur Brothers. Fig. 16 was taken near the close of Winston & Co.'s work, August 15, 1896, and shows the quarry- and retaining-wall derricks in operation. Winston & Co. also had the contract for the 2700 cubic yards of retaining-wall on Section 8.

Section 3.—In the re-advertising of Sections 2, 3 and 4 and the proposition submitted by McArthur Brothers, the bid of Gilman & Co. for Section 3 was the lowest. On January 3, 1894, McArthur Brothers relinquished the contract; on the 10th the section was formally awarded to Gilman & Co. Their glacial-drift plant was similar to that for Sections 2 and 4; and their rock-plant consisted of 4 Lidgerwood cable-ways. Drilling was done with compressed air, supplied by a Rand 18 by 30 duplex air-compressor. Drilling on this section cost 6 cents per cubic yard; channeling, 7 cents per cubic yard; blasting, 9 cents per cubic yard; and pumping, 2 cents per cubic yard.

Section 5.—This section was assigned January 18, 1894, to the Qualey Construction Co., which was working the section at the time as sub-contractor. Six methods were used for the excavation of the glacial-drift, as follows: wheelbarrows; wheel scrapers; car-incline hoist (cars loaded by hand); clam-shell derricks; steam-shovel, locomotive and cars; and steam-shovel working in connection with car-hoist. For the rock-excavation, a Lidgerwood cable-way and two McMyler revolving derricks were employed. Fig. 17, which shows the latter, is interesting also as an excellent instantaneous view of a blast. The McMyler derricks were moved from Section 7 after the completion of that section in May, 1896. Some of the rock was taken out with the clam-shell derricks, which were arranged to handle that material. Car-incline hoists were also used to some extent for this work. This section has had two serious land-slides, one in March, 1894, and the other in September, 1896. Both were caused by the weight of the spoil-banks placed upon the berm, which, owing to the plastic or quicksandy nature of the material overlying the rock, would not sustain the additional load. The first catastrophe occurred

before the retaining-wall was built, and the second after the completion of the wall and the canal at that point. In each case the slide necessitated the removal of over 16,000 cubic yards of material.

Section 6.—This section was assigned to Mason, Hoge, King & Co., April 25, 1894, and was considered one of the hardest and most discouraging sections on the canal; but under the skillful management of Mr. John King it turned out well for the contractors. Charles Vivian & Co., sub-contractors, excavated some 60,000 cubic yards of the top-soil and clay overlying the hard material on the east end of the section by means of a steam-scraper, which was operated with cables and a winding-engine. The device worked fairly well in easy-digging material, free from boulders, but would not work in the hard material.

The west half of the section extended through a marsh adjacent to the Des Plaines river. The depth of the vegetable and alluvial deposit was from 8 to 18 feet, and for its removal Charles Vivian designed a hydraulic dredge, which proved to be a splendid success. It excavated 241,000 cubic yards in 251.8 10-hour shifts, actual working time, which was at the rate of 957 cubic yards per 10 hours. The best month's record made by this dredge was 56,061 cubic yards, and the best day's record 2740 cubic yards in two 11-hour shifts. The cost of excavation was 4.15 cents per cubic yard, including repairs and maintenance, exclusive of cost of plant. Including cost of plant, it was 10.08 cents per cubic yard. Of the total volume moved by the dredge, as given above, 28,000 cubic yards were excavated from Section 7. After the dredge-work was completed, this part of that section was unwatered by Mason, Hoge, King & Co., and the cemented gravel, boulders and glacial drift were taken out with Little Giant traction steam-shovels, loaded into cars, hauled by mules to the foot of incline-hoists and, when raised to the top of the berm, again hauled by mules to the dump. The rock-plant consisted of 4 Lidgerwood travelling cable-ways, which were also used in the excavation of some of the glacial drift. A 20 by 30 duplex Rand compressor supplied compressed air for the drilling, hoisting cars on incline, pumping and working of derricks.

Section 7.—On April 25, 1894, this section was assigned to

Mason, Hoge & Co., and was worked by Gooch, Rinehart & Co., members of the Mason-Hoge syndicate. This section is the beginning of the all-rock sections, which did not require any special plant for the removal of the glacial drift, other than wheel or drag scrapers, wheelbarrows, and mules and carts, which were the means employed for the limited amount of earth on this as well as other sections following. The rock-plant consisted of three car-incline hoists for the top-lift, and two McMyler revolving derricks (Fig. 19; see also Fig. 14), placed in the pit and delivering the rock to the dump.

About the time that Gooch, Rinehart & Co. took charge of the work, the McMyler system was changed. The revolving derricks were taken from the pit and placed upon the berm, the boom being lengthened from 82 feet to 123 feet, giving a radius of 97 feet. Two of these derricks worked together, opposite each other; that is, one was placed on each side of the channel, and each removed half of the excavation, delivering it direct from the pit to the spoil-bank. It is believed that these machines hold the banner record over all others for the greatest 10-hour output, which was made on March 18, 1895, when two of these derricks removed 605 bucket loads, giving a total volume of 980 cubic yards of solid rock-working. There were 59 laborers in the pit, making the amount of rock handled 16.6 cubic yards per man. These derricks removed 279,323 cubic yards in 492 10-hour shifts, giving an average of 568 cubic yards per shift for the entire period. After the derricks had been changed to independent machines, the two conveyors were re-built into cantilevers (Fig. 18), one arm extending to the center of the canal and the other arm 105 feet over the spoil-area, the total length of the cantilever being 231½ feet. These cantilevers, like the derricks, travelled on tracks parallel to the canal. The volume of rock handled by the cantilevers was 168,475 cubic yards in 337 10-hour shifts, which was equivalent to 500 cubic yards per day, for the two machines. A Lidgerwood cable-way completed the rock plant. All the drilling and some of the drainage-pumping and hoisting was done with compressed air, supplied by a 20 by 30 duplex Rand compressor.

Section 8.—This section was assigned to Mason, Hoge, King & Co. on January 18, 1894. Two-thirds of the section was excavated

by Mason and King, and one-third by Rosser, Hoge and Scruggs, all members of the Mason-Hoge syndicate. In the excavation of the top lift or stope, car-incline hoists of the Mason-Hoge type were used (Fig. 21). Mules and carts were also used to some extent in this part of the excavation. From five to six cars were worked at each face. The cars, when loaded, were drawn one at a time to the foot of the incline and hoisted to the top of the spoil-bank, which was from 18 to 25 feet high. Here again the cars were hauled by mules over diverging tracks to the points of dumping. Frequently two faces were worked by one incline. The efficiency of this method was as high as 575 cubic yards per 10 hours; this being an average for a month. The average haul in the pit and on the dump was 600 feet.

The two remaining lifts were removed with 5 Lidgerwood cable-ways. All drilling, pumping and hoisting on this section was done with compressed air. The section is now finished with the exception of 19,782 cubic yards of rock at the point where the Santa Fe R. R. crosses the channel. The greatest output on this section for any one month was in June, 1894, when 72,934 cubic yards of rock and 3516 cubic yards of earth were excavated.

Section 9.—This section was assigned to Halvorson, Richards & Co., January 18, 1894. They assumed and increased the plant which had been installed by a sub-contractor under their predecessors. This plant consisted of two Hulett-McMyler derricks (Fig. 19) and conveyors, which were identical with those placed on Section 7. These machines on Section 9 were not remodelled or reconstructed into four independent conveyors, as was the case on Section 7, but were used as first built until the completion of the section in October, 1895. The total number of cubic yards of rock handled by these two machines was 318,979 in 401 days, giving an average output of 398 cubic yards per day per machine. In addition to the McMyler machinery, four car-inclines were used, not only for the removal of the first lift, but also for the second and third lifts. This was the only section on which car-inclines were employed in connection with the rock excavation on the second and third lifts; and this was a distinctive feature of the work on this section. The spoil-dump began at the 50 feet berm-limit and extended

back to the river-diversion levee on the north side of the canal, a distance of about 1000 feet. The average haul was 700 feet. Trains of ten and twelve 1½-"place"-yard capacity Petler dump-cars were hauled by teams to and from the working-face to the foot of the incline, and after being hoisted to the level of the dump were again drawn by teams to the point of dumping. The efficiency of these inclines was from 450 to 600 cubic yards per shift. This was the only one of the rock-sections which did not use compressed air for the drilling.

Section 10.—This is one of the long sections of the canal, being 6000 feet in length. The first shovelful of earth was taken from this section and the first blast fired on the canal took place here. The event was appropriately observed by "Shovel-Day" exercises, September 3, 1892. Just three years after this, on September 3, 1895, Section 10 was completed—first of all the sections of the canal. This event was likewise commemorated with suitable ceremony. A tablet bearing the two dates, etc., was placed in the side of the rock on the dividing-line between Will and Cook counties.

E. D. Smith & Co. were the contractors for this section. The top-lift was removed with car-inclines of a special design, used only on this section. Briefly described, it consisted of a single track on the dump, which, at a point about 75 feet from the edge of the canal, branched into two tracks forming a loop in the pit. Each incline was equipped with 12 side-dump cars, and the loaded cars were hauled out in trains of three cars by a hoisting-engine, placed some distance back on the spoil-bank. The cars, when dumped, returned to the pit by gravity, the speed being controlled by the hoist-engineer. The empty trains returned to the pit over the loop. The advantages of this type of car-incline were: a minimum haul, and no delays due to men waiting for cars. Three of these inclines were employed and some 315,000 cubic yards solid rock were removed by this means. The average efficiency of this type of incline was about 300 cubic yards per day. The triangular portion of rock on the side of the canal over which the incline passed to the berm was removed by mules and carts after the incline-excavation had been finished. The remaining excavation, consisting of the second and third lifts, was removed by three Brown cantilevers (Fig. 20). These machines had a total length of 355

feet. The height of the arm extending over the dump was 95 feet above the ground, and allowed a spoil-bank 80 to 85 feet in height. The crane was mounted on trucks and moved back and forth on the tracks along the berm by gearing connected with the engine. In moving along the berm, the speed could be varied from 150 to 400 feet per minute. Three levers controlled all movements of the machinery. The total weight of each cantilever was 150 tons, and the cost was $28,000. The travel of the bucket was 343 feet, and the load was dumped automatically at any desired point on the upper arm. The contracts for the first two of these machines placed on the canal called for an efficiency of 225 bucket-loads per day, which was equivalent to about 390 cubic yards of rock. But after the first year these machines reached a monthly output of 15,000 to 16,000 cubic yards per cantilever, which was equivalent to 600 cubic yards of rock per 10 hours.

Two 18 by 30 duplex Meyer-valve Rand air-compressors furnished compressed air for all drilling, pumping and hoisting.

Sections 11, 12 *and* 13.—These sections were under one contracting firm, Mason, Hoge and Co., and will be described together for that reason, and also because the methods employed in the excavation were substantially the same, though the work was divided among a number of subordinate companies formed by the members of the firm named. The top-lifts were taken out with the Mason-Hoge car-incline (Fig. 21). Carts and mules were to some extent used for the first stope. After the first lift had been excavated, eight Brown cantilevers were installed for the remaining excavation. These machines began work about October 10, 1893, and finished November 6, 1895, working 3608.7 10-hour shifts, and moving 2,084,700 cubic yards solid rock at a cost of 2 cents per cubic yard. This included only the cost of conveying the rock from the pit to the dump, and did not cover the cost of channeling, drilling, blasting, loading rock into the buckets or superintendence. The cost of conveying (2 cents per cubic yard) included the cost of labor operating the cantilever, fuel, oil, waste and repairs. The Brown Hoisting and Conveying Machine Co. of Cleveland, Ohio, erected and operated these machines, receiving from the contractors 15 cents per cubic yard, which gave them a clear profit of about 13 cents per cubic yard. One of the first Lidgerwood cable-ways placed on the canal was on Section 11, and

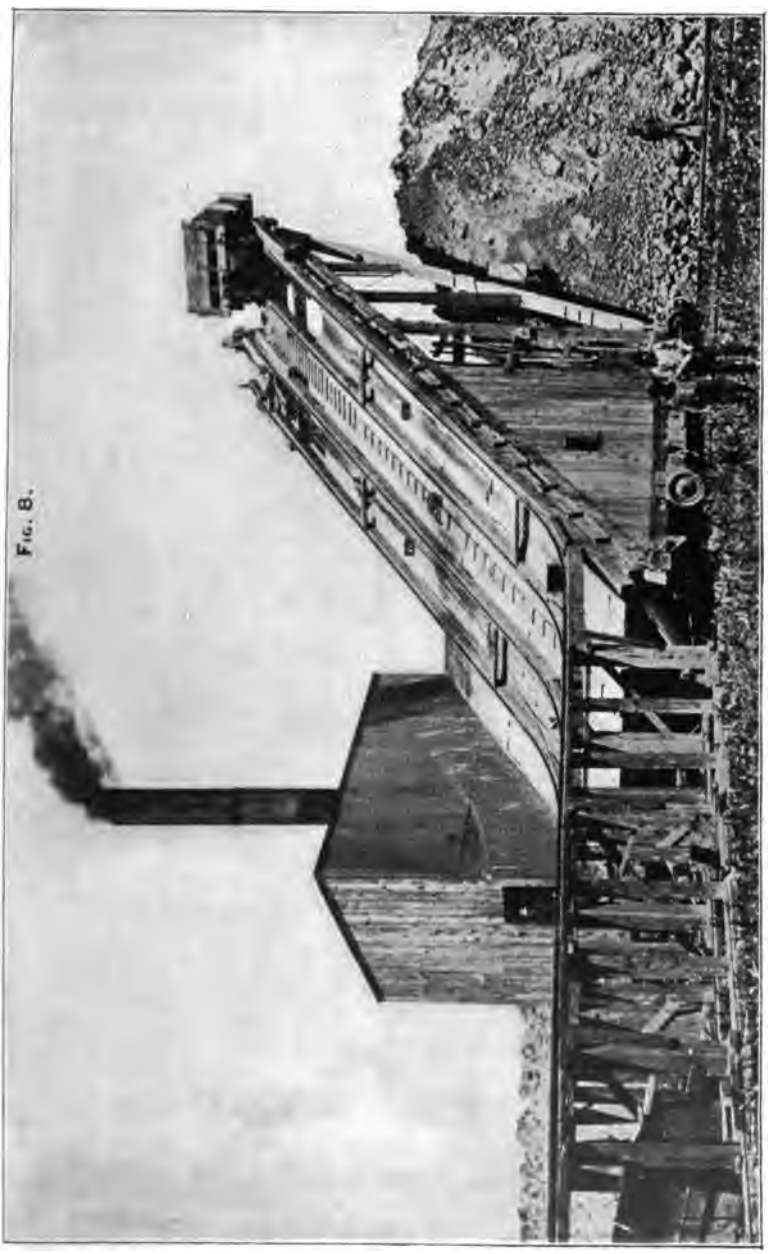


FIG. 8.

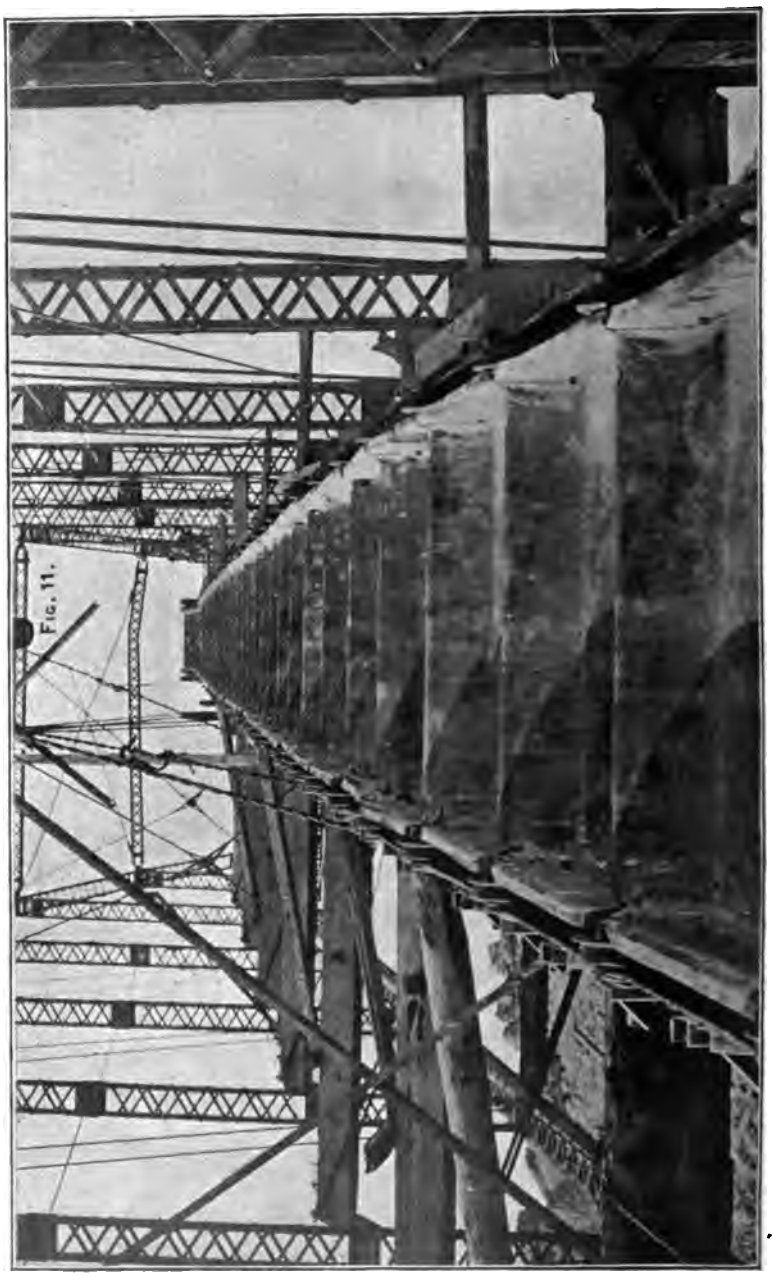
Sections L and M.—Heidenreich Incline and Tipple.

FIG. 9.

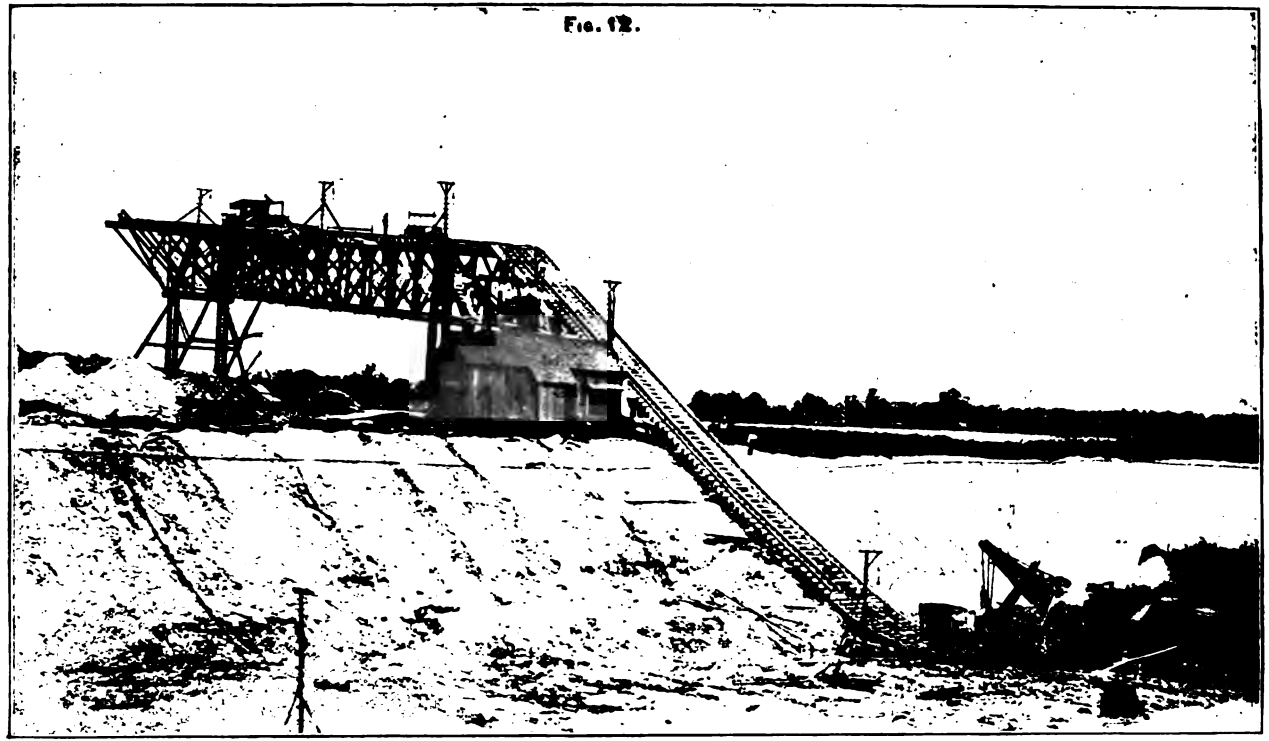
Sections I and K.—Christie & Lowe's Bridge-Conveyor. (Excavation made in two benches, and canal finished as work advanced.)

FIG. 10.

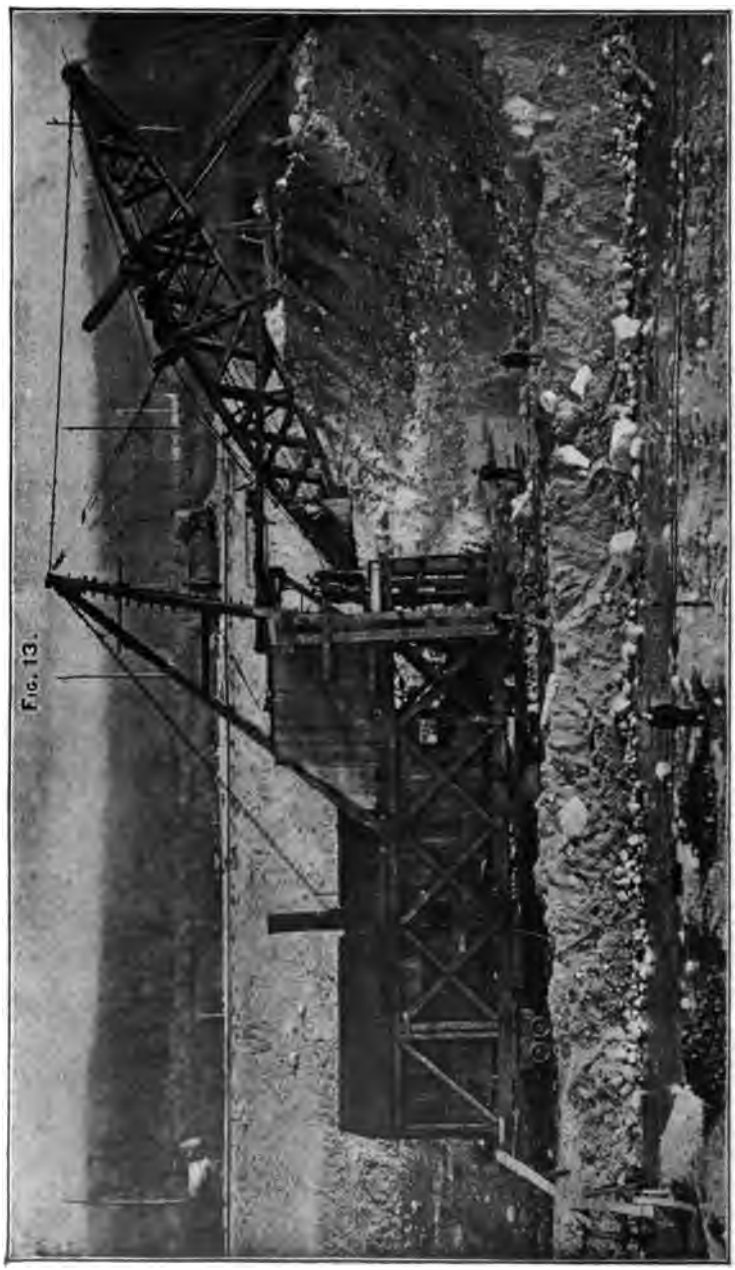
Section H.—General View of Hoover & Mason Steel Belt Excavator (see Fig. 11). (Canal finished to grade.)



Steel Belt of Hoover & Mason Excavator (see Fig. 10). (Looking up the Cantilever-arm towards the spoil-bank.)



Section A.—Gould Conveyor.
(The steam-shovel is just "cutting out," *i.e.*, finishing one of the transverse cuts across the channel from slope to slope.)



Section A.—Jones-Lovell Land-Dredge, Used in Revetment and Slope Work ahead of Gould Conveyor (Fig. 12).

Sections 2, 3 and 4.—Lidgerwood Traveling Cableway. (This view was taken on Section 7, where this cableway was also employed. On the right is seen a Hulett-McMyler revolving derrick.)

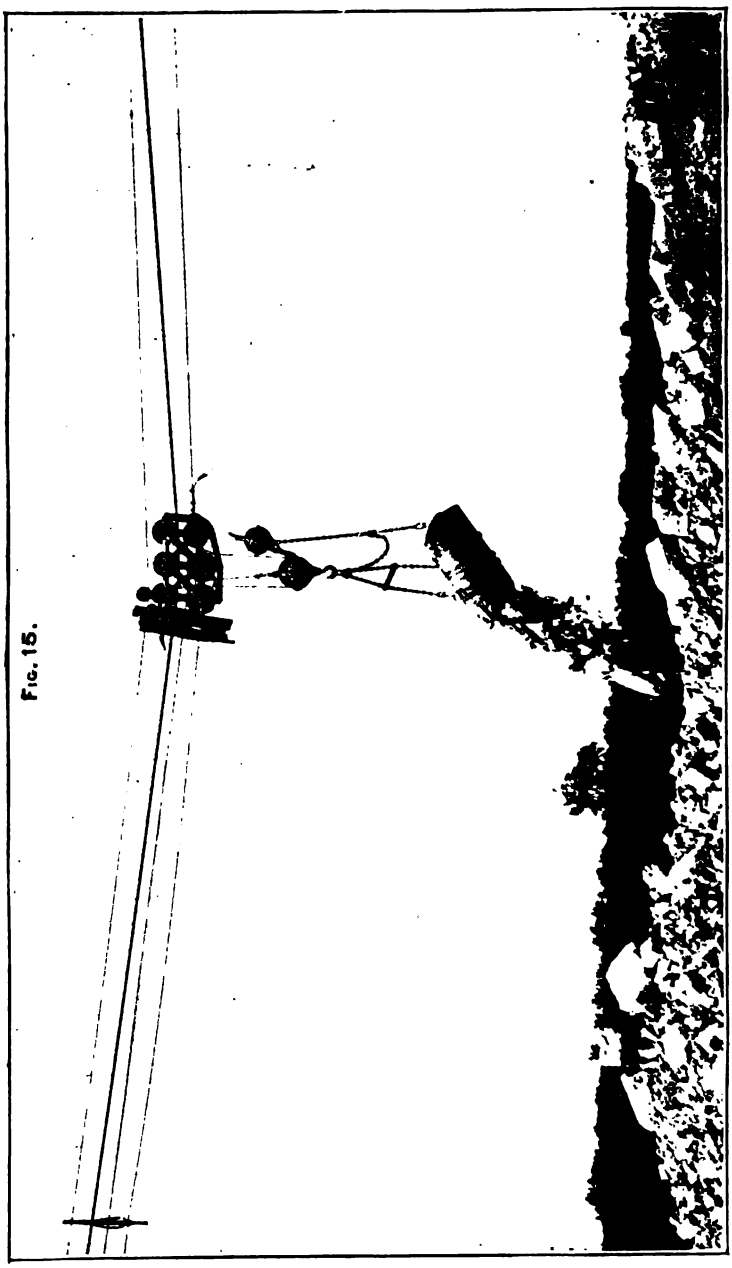
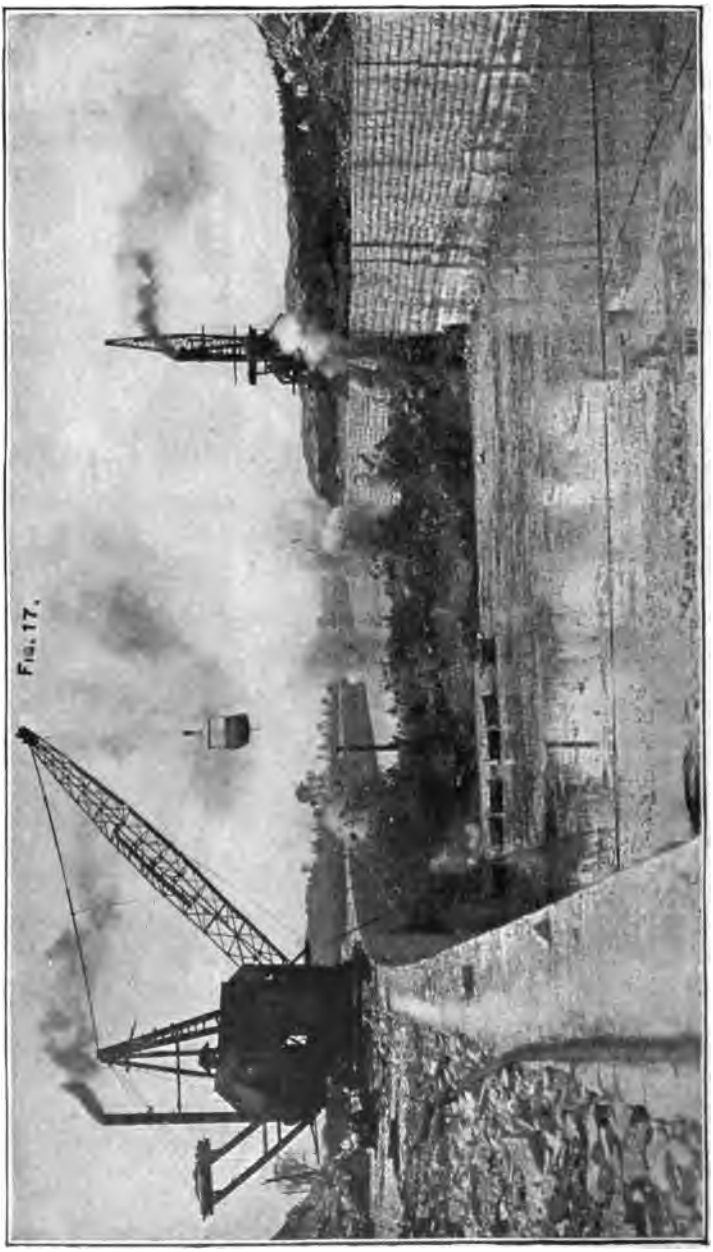


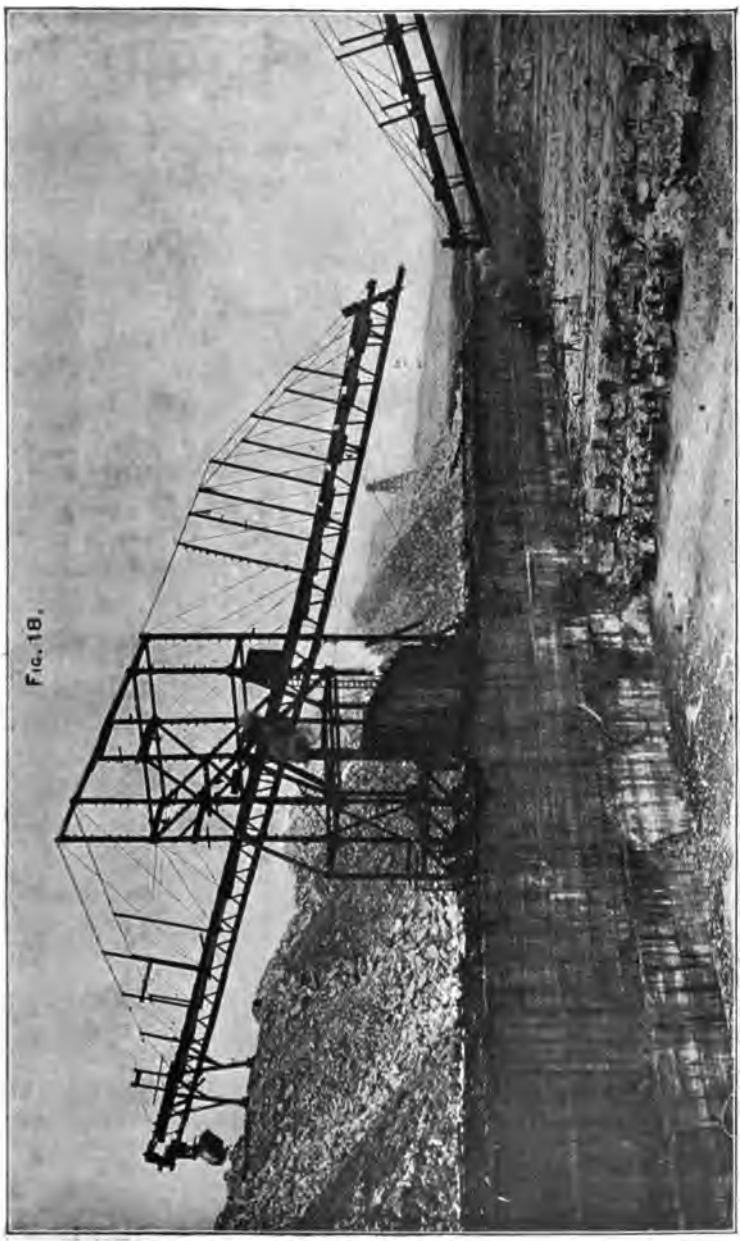
FIG. 15.

Locke Patent Aerial Dump, Used on Lidgerwood Traveling Cableway.

Section 2.—General View of Winston & Co.'s Retaining-Wall and Quarry Work, at its Close, Aug. 15, 1896. (In the foreground a Lidgerwood Cableway is removing the débris remaining from the wall-work.)



Dynamite-Blast, under Hulett-McMyler Revolving Derrick. (Looking west on one of the curves. A charge of 400 lbs. of dynamite is displacing about 600 cubic yards of solid rock.)



Section 7.—Hulett-McMyler Cantilevers. (One on either side of the canal, and each serving about one-half of the excavation.)

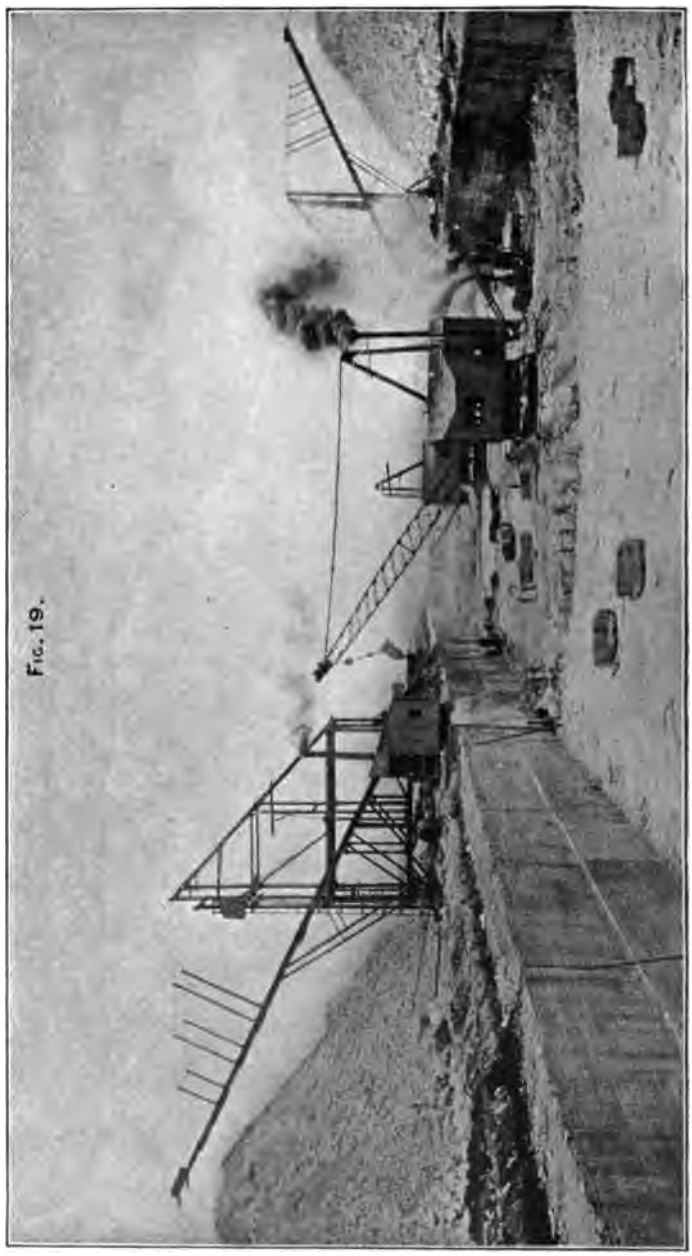


FIG. 19.

Section 9.—Hulett-McMyler Derrick and Conveyor. (Working on second lift or bench.)

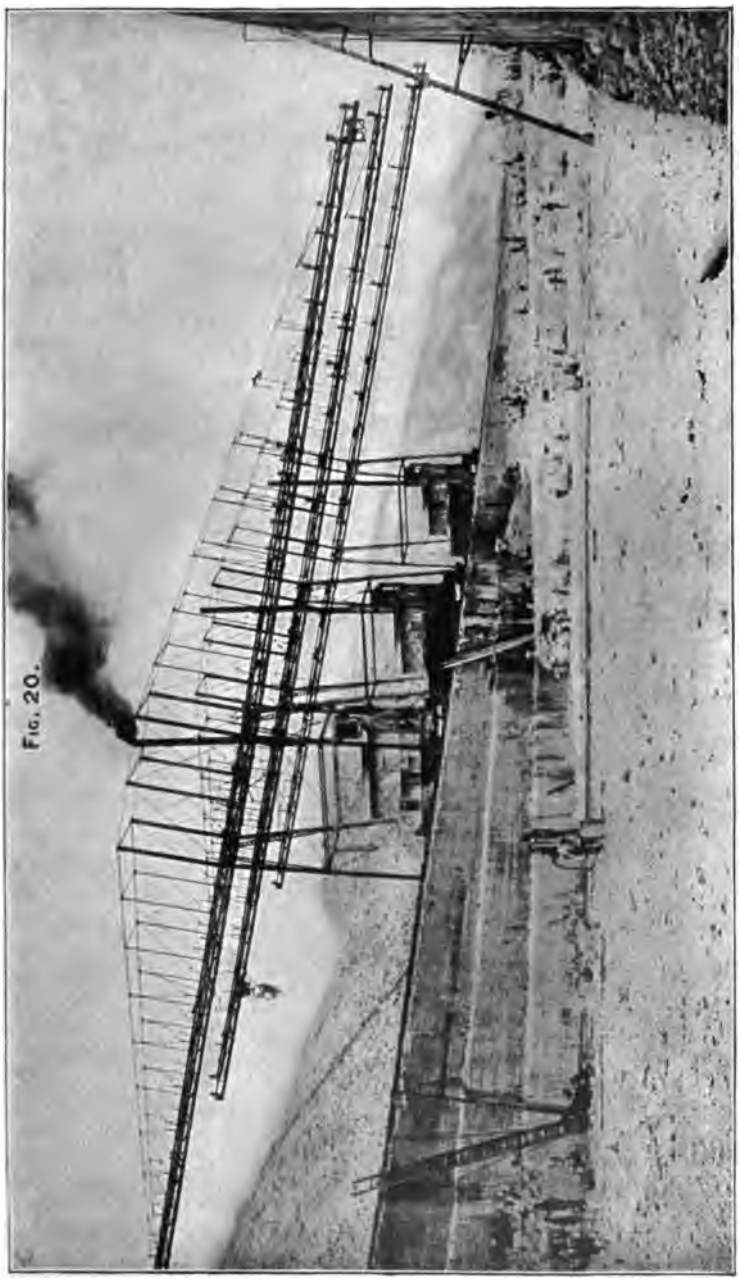


FIG. 20.

Section 10.—Three Brown Cantilevers. (Canal finished in the foreground.)

FIG. 21.

Section 12.—Mason-Hoge Car-Incline. (At work on upper bench.)

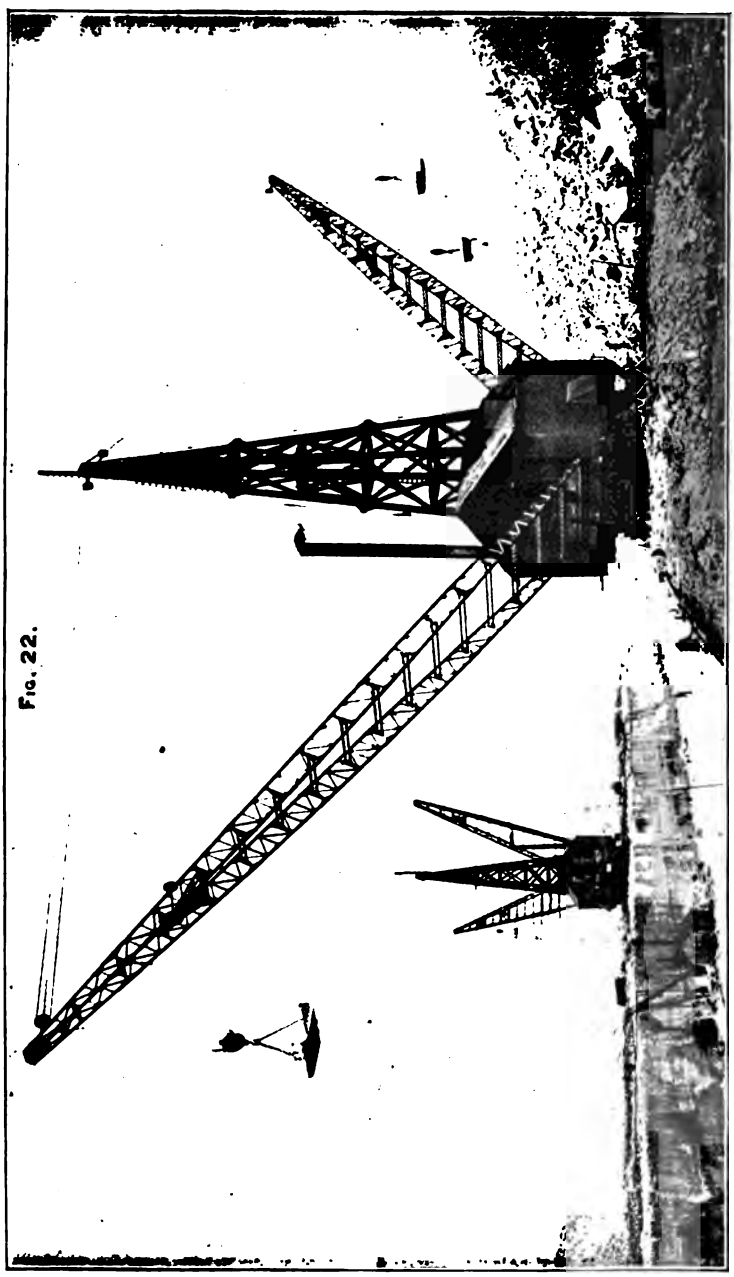
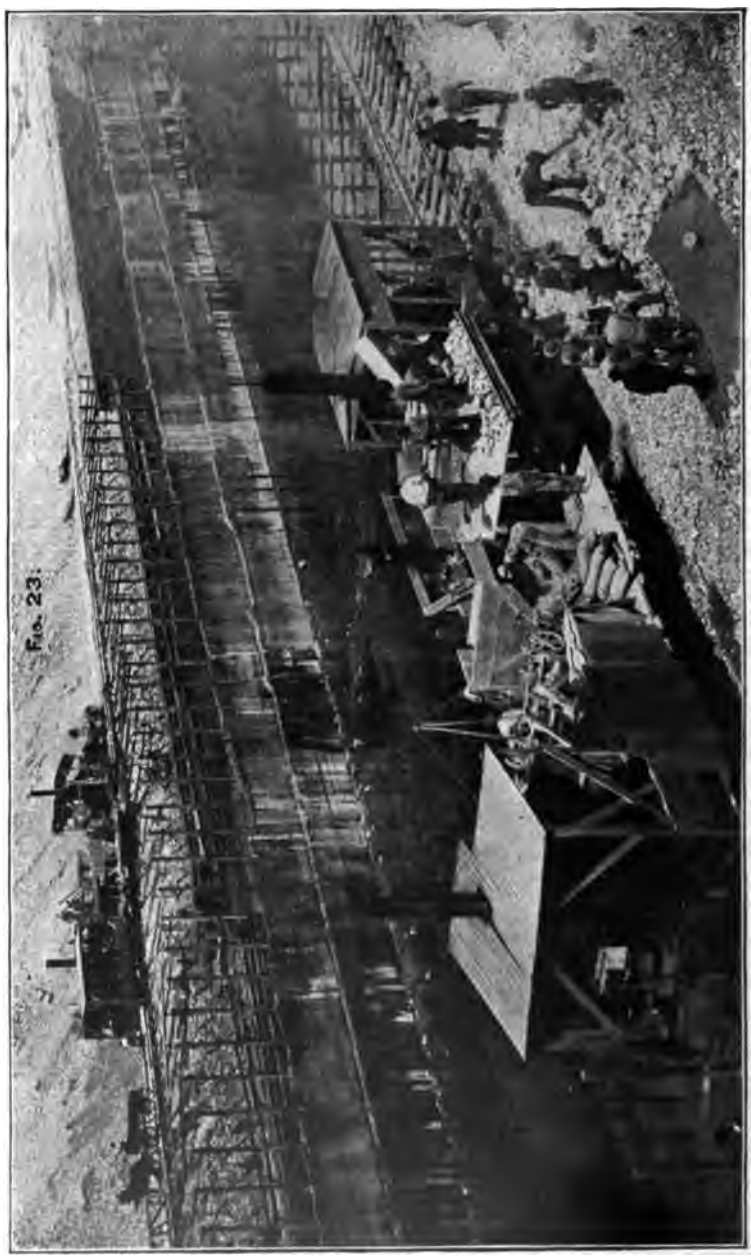


FIG. 22.

Section 14.—Geraldine Double-Boom Self-Contained Revolving Derricks. (Length of each boom, 150 feet.)



Section 14.—General View of Concrete Retaining-Wall Work, Showing Combination Stone Crushers and Concrete Mixers. (The machines in the foreground and background respectively are at practically the same level, on opposite sides of the channel. The view is taken from the top of the rock spoil-bank, looking down.)

FIG. 24.

General View of Controlling-Works. (Sluice-gates in position.)

it was upon this machine that Mr. Chas. H. Locke invented the aerial dump (Fig. 15), which made the use of the cable-way practicable on the drainage canal. Drilling, drainage-pumping and incline-hoisting on these three sections was done with compressed air.

Section 14.—This section, awarded January 15, 1894, to Smith and Eastman, passed through what is known as "the Fitzpatrick mound," which necessitated an average cutting of 18 feet in the drift before the surface of the rock was reached. The total volume of glacial drift on this section was 379,380 cubic yards, and there were 1,023,500 cubic yards of solid rock, and 22,000 cubic yards of concrete retaining-wall. The plant installed by Smith and Eastman for glacial drift or earth was the steam-shovel, locomotive and car system; and the rock-moving plant consisted of 4 Geraldine double-boom self-contained revolving derricks (Fig. 22) and 2 double-boom fixed or guy-derricks. The first were designed by Dion Geraldine, of Chicago, and the guy-derricks by the American Hoist and Derrick Co., of St. Paul, Minn. The revolving derricks each had two booms 150 feet long. The two fixed derricks were placed on each side of the canal and directly opposite each other, and were guyed together. The average output of the Geraldine derricks was about 300 to 350 cubic yards per day, and that of the St. Paul derricks 150 to 200 cubic yards per 10-hour shift. The Geraldine derricks were moved along the berm on wooden rollers, as the excavation advanced, passing over planks or heavy timbers in much the same manner as a building is moved.

The glacial drift excavated was used in building a levee which extended the entire length of Sections 14 and 15 on the west side, and on the east over part of Section 14 and all of Section 15. This embankment is 30 feet wide on top and 8 feet above Chicago datum. Section 14 has the honor of having the greatest solid rock-output on the canal for any one month. This record was made in April, 1895, when 86,200 cubic yards were excavated. The output for the next month, May, was 81,500 cubic yards. No satisfactory agreement for the building of the retaining-wall was reached between the District and Smith and Eastman; and the contract for this work was finally let to Campbell, Dennis & Co.

Owing to the poor quality of the rock on Sections 14 and 15, the retaining-wall was built of concrete masonry. The machines for this work are shown in Fig. 23. Each of these machines was a combination stone-crusher and cement-mixer, which crushed the stone, mixed the sand, cement, crushed stone and water, and delivered it to the wall cast or form. The machines moved along the berm on tracks. Compressed air was used for the drilling on this section.

Section 15.—The contract for this section was awarded to Wright, Meysenburg, Sinclair and Carry, August 29, 1894. The prices for the work were unusually low, being 19 cents for glacial drift, 59 cents for solid rock and $2.35 for rubble retaining-wall, which was raised to $3.40 per cubic yard when the specifications were changed so as to require concrete wall. In determining upon a plant for this excavation, the contractors had to take into consideration the following requirements: (1) that no spoil could be deposited higher than 8 feet above datum; (2) that the spoil could not be deposited on the lower 1000 feet of the channel; and (3) that no material could be deposited on the south side of the channel. The plant decided upon was the steam-shovel, locomotive and car system.

The concrete retaining-wall was built in much the same manner as just described on Section 14, the only difference being that the stone-crusher was separate from the concrete-mixing machine. Drilling and pumping were done with compressed air.

Controlling-Works.—The discharge from the drainage canal into the tail-race leading to the Des Plaines river will be controlled by a series of sluice gates (Fig. 24) and a bear-trap dam. These gates are of riveted steel and weigh 15 tons each. There will be 7 gates in actual use and 8 "blind" gates, which are for use in case of some future emergency.

I wish to make special acknowledgment to Mr. Isham Randolph, chief engineer of the drainage canal, and also to Mr. William M. Christie, civil engineer, of Lemont, Ill., for valuable assistance and information, furnished to me in the preparation of this paper.

The Fullers' Earth of South Dakota.*

BY HEINRICH RIES, PH.D., NEW YORK CITY.

(Lake Superior Meeting, July, 1897.)

FULLERS' earth is a clay-like substance, which has the property of decolorizing or clarifying oils. An ultimate chemical analysis shows it to differ from most ordinary clays in having usually a high percentage of combined water and a low amount of alumina. There is probably a large amount of hydrous silica present. Fullers' earth possesses little or no plasticity, and in order to work properly has to be ground very fine. A chemical analysis is of little value at present in determining its quality; only a practical test suffices.

Up to about two years ago all of the fullers' earth used in this country was imported from England; but about that time deposits were discovered in Florida,† and since then rumors of its occurrence have been coming from all parts of the country.

Next to the Florida deposits of fullers' earth, those of South Dakota are the most extensive thus far discovered in this country.

The earth was first discovered near Fairburn, Custer county, and other deposits have since been prospected at Argyle, Minnekahta and other localities.

Fairburn.—One of the first deposits located and opened up is five miles southeast of Fairburn, Custer county. The mine is a mile from the Fremont, Elkhorn and Missouri Valley Railroad, and the section exposed shows, beginning at the top:

	Feet.
Micaceous sandy clay, ...	6
Fullers' earth, ...	9
Micaceous sandstone, ...	–

The fullers' earth passes upwards into the sandy clay, but in its purer portions it is a yellowish, gritty clay, with a somewhat

* Published by permission of the Director of the U. S. Geol. Survey.

† 17th Ann. Rep. U S. Geol. Sur., Part iii., page 877.

nodular structure. The individual nodules are dense, and break with a conchoidal fracture. On account of their density and hardness they remain intact when washed down the slope from the outcrop, and are therefore a guide to the prospector.

The mine from which the above section is given belongs to the Mining Syndicate Company of Omaha. A few tons have been taken out. Before shipment, the earth is spread out to air-dry on a wooden platform which has been erected at the mine.

A test-shaft has been sunk in fullers' earth about two miles north of Fairburn. The earth here is similar to that at the previous mine, but is associated with a pitted limestone. There seems to be an abundance of the material.

Argyle.—Two miles west of Argyle, in Township 6 S, R 3 and 4 E, an opening has been made in the hill-slope disclosing a bed of fullers' earth of considerable thickness. A shaft was first sunk to a depth of 26 feet, and from it the following section was obtained:

	Feet.
Surface pebbles,	1
Sandstone,	1½
Fullers' earth,	18
Clay and earth,	2
Sandstone,	5

An excavation has been made about 50 feet southeast of this shaft, and 8 feet of fullers' earth is shown. It also crops out half a mile farther north. The material is of exactly the same character as that near Fairburn.

Minnekahta.—One mile and a quarter west of the railroad station is a bank of Jurassic shale 40 feet thick, capped by red sandstone. It is a fine-grained soft shale, totally different in character from the clay or fullers' earth at Argyle and Fairburn. This mine was first opened in June, 1896, and up to October, 1896, six to eight carloads have been shipped. If the earth proves to have the proper clarifying qualities, there is any desired quantity of it to be had.

The Fairburn and Argyle earths have been tested with cotton-seed oil, and have given excellent results.

Other Localities.—Since the discovery of the fullers'-earth beds at Fairburn, people have been staking out claims all over the Black Hills region and south of it; but many of these

will probably turn out to be nothing more than common clay. Among these minor localities may be mentioned the points nine miles west of Custer City, two miles south of Buffalo Gap, eight miles west of Edgemont. These are mostly prospects. Up to October, 1896, no earth had been shipped from them.

Analyses.—The following analyses, compiled from various sources, serve to show the usual composition of the South Dakota fullers' earths:

	1.	2.	3.	4.	5.	6.
Silica,	68.23	60.16	56.18	55.45	57.00	58.72
Alumina,	14.93	10.38	23.23	18.58	17.368	16.90
Ferrous oxide,	3.15	14.868	1.26 Fe_2O_3	3.82	2.632	4.00
Lime,	2.93	4.96	5.88	3.40	3.00	4.06
Magnesia,	0.875	1.714	3.29	3.50	3.027	2.56
Loss on ignition,	6.20	7.20	11.45 H_2O	8.80	9.50	8.10
	96.31	99.282	101.29	Vol. 5.35	5.85	Alk. 2.11
				98.90	98.377	Moist. 2.30
						98.75

1. From William Bodenner's pit, north of Fairburn.

2 and 3. From M. Palmiter, Fairburn.

4 and 5. From D. Henault, Custer City.

Analyses 1 to 5 were made by Prof. Flinterman, of the South Dakota School of Mines, at Rapid City, and kindly furnished by Prof. F. C. Smith.

6. From southeast of Fairburn. E. J. Riederer, analyst.

The South Dakota fullers' earth has thus far been used chiefly in Omaha, and also somewhat in Chicago. Samples from Fairburn, which were tested, bleached fully as well as the English earth.

With the further development of these deposits there is no reason why the South Dakota fullers' earth, provided it is all of proper quality, should not supply many lard and oil factories in the West. There are lard companies in Denver, for example, which are now using the English earth. There are also petroleum refineries in which the Dakota material might be introduced.

A plant for pulverizing the earth has been erected at Hot Springs, and thus located it can receive the crude material from either the Fremont, Elkhorn and Missouri Valley, or the Burlington and Missouri R. R.

The Clays and Clay-Working Industry of Colorado.*

BY HEINRICH RIES, PH.D., NEW YORK CITY.

(Lake Superior Meeting, July, 1897.)

THE clay-resources of Colorado are but little known, for the reason that little has been published concerning them, and also on account of the hitherto small demand for clay products in the Rocky Mountain region.

At the present day the wares manufactured from clay which are used in Colorado include common and pressed brick, terra-cotta, sewer-pipe, pottery and refractory articles. Most of these are brought from a considerable distance; but with an increasing demand for such goods in the West and a better knowledge of the available materials, there should be considerable expansion in the clay-working industry of Colorado in the future.

The notes here presented are an abstract of those collected by the writer in October, 1896, in a reconnaissance of the clay-bearing formations of Colorado.

Clay-bearing Formations.

We may roughly divide the clay-bearing formations of Colorado into three groups:

1. Loess and alluvial deposits.
2. Jura-Trias and Cretaceous clays.
3. Tertiary clays.

The loess forms an extensive deposit of large area, though not very thick, extending from north to south across the State, and eastward from the foot-hills. It is generally a very sandy clay, with little plasticity, and is underlain by a gravelly stratum, which determines the workable depth. Clay similar to the loess is found underlying the river terraces in many of the broader valleys, such as those of the Arkansas, Grand River, etc.

The Mesozoic formations extend along the eastern edge of the Rocky Mountains, and also occur in some of the deeper valleys tributary to the foot-hill belt. They consist of a great series of interbedded shales, sandstones and limestones of Jura-

* Published by permission of the Director of the U. S. Geol. Survey.

Trias and Cretaceous age, the beds being tilted at a high angle. The Jura-Trias shales have not been utilized. The Cretaceous beds which overlie them consist of the following divisions:*

1. The Dakota shales and sandstones.
2. The Benton shales and limestones.
3. The Niobrara limestones.
4 The Pierre shales.
5. The Fox Hill shales.

The yellowish gray Dakota sandstones are usually interbedded with shales and clays, which are often highly refractory in their character. The shales are frequently arenaceous, and both the clays and shales sometimes contain enough vegetable matter to color them brown or black. The Benton group carries great quantities of shale and a central limestone member. The upper, or Carlile, shale is of importance to brick manufacturers. The Pierre and Fox Hills group both carry shales, but they are not utilized.

The Tertiary clays or Denver beds carry great deposits of clay, and are developed east of Golden and Boulder.

Brick-Clays.

All of the common brick manufactured in Colorado are made either from the loess or from the river-clays in the valleys. The clays are usually lean, coarse-grained and gritty, and possess little plasticity.

A sample of loess from one of the Denver brick-yards was examined physically with the following results:

The clay was porous, and easily broken. When mixed with water, 14.5 per cent. was required to give a workable paste; that is, one of the proper consistency to permit moulding. The percentage of water thus absorbed is low, as is often the case with lean clays. Bricklets moulded from this paste shrunk 6 per cent. in drying and an additional 3 per cent. in burning. Briquettes (of the same shape as those made in testing cement) made from this clay had, when *air-dried*, an average tensile strength of 55 pounds per square inch and a maximum tensile strength of 66 pounds per square inch, which is also low, owing to the lean nature of the clay. At a temperature of 2000°

* G. K. Gilbert, "The Underground Water of the Arkansas Valley in Eastern Colorado."—*U. S. Geol. Survey 17th Ann. Report*, Part ii., p. 561.

F. the clay showed incipient fusion; at 2100° F. it vitrified and became practically non-absorbent, and at 2200° it fused completely. At 2000° F. it burned red; but above that temperature the color deepened rapidly.

When under-burned, the brick is porous and weak, and very similar to the product seen at many of the smaller yards.

The loess brick-clays are most extensively worked around Denver, while other less important localities are Colorado Springs, Pueblo, Trinidad, Boulder, Loveland, Greeley, etc. The river-terrace clays are worked at Glenwood Springs, Florence, Aspen, Grand Junction, etc. In the case of these latter the less sandy clays are more apt to be found back from the river.

Pressed-Brick Clays.

The Cretaceous and Tertiary formations of Colorado contain an abundance of clay suitable for the manufacture of pressed brick. Three localities will be mentioned as representing different formations.

The first is at Golden, where a red, decomposed shale is used by the Golden Pressed and Fire-Brick Company. This shale mellows to a fine-grained, gritty clay of moderate plasticity.

The sample tested required the addition of 22 per cent. of water to make a workable paste, that shrunk 8 per cent. in drying and 4 per cent. in burning. The average tensile strength of the air-dried briquettes was 55 pounds per square inch, with a maximum of 67 pounds.

Incipient fusion occurs at 1850° F., vitrification at 1950° F. and viscosity at 2050°—2100° F. The clay burns to a bright red, and makes a good dry-pressed brick.

At Boulder a brownish gray, hard, gritty shale occurs in abundance, and also gives good results with the dry-press process.

The third locality is at La Junta, where the Carlile shale is utilized for dry-pressed brick. This shale not only makes a good brick, but on account of its softness is easily prepared. Its wide-spread occurrence in the West may prove of importance in the future.

A sample of this shale gave, on the addition of 30 per cent. of water, a workable paste of high plasticity, which shrunk 13 per cent. in drying and 2 per cent. in burning. The air-dried

briquettes had an average tensile strength of 145 pounds per square inch and a maximum of 156 pounds. Incipient fusion occurred at 1800° F., vitrification at 1950° F. and viscosity at 2100° F. The shrinkage in drying is large, and the clay has to be dried slowly to prevent cracking; but with the dry-press process the air-shrinkage is of little importance.

Fire-Clays and Pottery-Clays.

These two grades of clay occur in close association, and interbedded with the Dakota sandstones, in the hog-backs extending along the eastern edge of the mountains. The fire-clay is generally represented by a dark gray or black fine-grained, shaly clay, which is extensively mined at Golden, and has also been worked at Parkdale and still more recently at Delhi. The fire-clay bed at Golden is 12 to 18 feet thick, and often faulted. About one-half of the stratum has to be rejected, as it contains too much iron, but the other half makes excellent fire-brick and smaller refractory articles for assayers' use.

The Parkdale fire-clay beds are on the average 4 feet thick, and more siliceous than those at Golden.

The pottery-clays, which are abundantly exposed south of Golden, are either sandy clays with little plasticity, which may at times grade into the associated sandstones, or else they are plastic clays, frequently charged with organic matter. A sample of the latter class which was tested showed it to be a fine-grained, gritty clay, requiring the addition of 31 per cent. of water to give a workable paste, which shrunk 12 per cent. in drying and 4 per cent. in burning. The tensile strength of the air-dried briquettes ranged from 160 to 164 pounds per square inch. Incipient fusion occurred at 2100° F., vitrification at 2250° F. and viscosity at 2400° F.

The best results are generally obtained by using a mixture of the plastic and sandy clays.

Clay-Products.

Common brick are manufactured in small quantities at many localities in the State. In many cases proper moulding and burning would render the product much stronger than it usually is. Pressed brick are only made at La Junta, Golden, Boulder and Denver. Terra cotta is not manufactured in Colorado, but

doubtless many of the tertiary clays would be found well adapted for this purpose.

Paving-brick have only reached the experimental stage, and their manufacture should be developed more. They are made from a mixture of fire-clay and tertiary clays.

There are three stone-ware potteries in Denver, but no white-ware is made, and it is doubtful if a white-ware pottery could be a financial success, as all of the raw materials would, from present indications, have to be brought a long distance.

For the manufacture of sewer-pipe the Laramie clays furnish an abundance of material, and there is one large pipe-works located at Denver.

The manufacture of refractory wares, such as fire-brick, locomotive blocks, muffles, scorifiers and crucibles has naturally been one of the important lines of the clay-working industry in the West, and the goods there made bear an excellent reputation. Four firms produce fire-bricks, and the smaller assayers' goods are made by two companies, one at Denver, the other at Pueblo.

The following table of analyses, taken from Part IV. of the 16th Annual Report of the U. S. Geological Survey (pp. 554–565), will serve to show the composition of several of the better grades of Colorado clays:

	1.	2.	3.	4.	5.	6.
Moisture, . .	47	1.36				3.75
Silica, ...	46.61	63.22	46.88	71.81	56.41	61.00
Alumina, . .	37.20	24.72	35.42	15.09	26.37	35.00
Ferric oxide, .	0.15	0.43	1.74*a*	1.75*a*	——	0.25
Lime, ...	0.44	0.30	0.44	0.14	0.29	——
Magnesia, . .	0.25	0.13	0.20	0.05	0.20	——
Alkalies, . . .	1.23	trace	1.19	1.02	1.55	——
Water, . . .	13.65	8.63	14.10*b*	10.14*b*	14.66*b*	
Org. matter, . .		.40				
Titanic acid, .		.68				
	100.00	99.87	99.89	100.00	99.48	100.00
Total fluxes, . .	2.07	0.86	3.57	2.98	2.04	

1. Edgemont, Jefferson Co. M. Moss, analyst.
2. Pueblo, Pueblo Co.
3. Golden, Jefferson Co.
4. Crucible clay, Denver Fire-Brick Co., Golden, Jefferson Co.
5. Kaolin, Golden, Jefferson Co.
6. Standard Fire-Brick Co., Pueblo, Pueblo Co.

a. Determined as ferrous oxide.

b. This includes moisture.

The Development of Lake Superior Iron-Ores.

BY D. H. BACON, SOUDAN, MINN.

(Lake Superior Meeting, July, 1897.)

NEARLY all men are too busy with their own affairs to keep informed of the progress in other lines. From time to time we read statistics showing the development in some industry that astonish us; but the feeling of surprise soon fades, to be followed by others that likewise pass away. A short résumé of what has been done in one line may be of interest.

Down to late in the fifties the ore-product of Lake Superior was handled over a mule-tram-road to Marquette; now the Duluth, South Shore and Atlantic, the Chicago and Northwestern, the Wisconsin Central, the Duluth, Superior and Western, the Duluth, Mesabi and Northern, the Duluth and Iron Range and the Lake Superior and Ishpeming railroads serve the shippers. Down to nearly 1870 a 700-ton ship was an enormous craft, the loading of which required two days; the unloading being seldom accomplished in that time. In 1871 the largest ore-barge carried 1050 tons and made twelve trips per season. Now the cargoes reach 5000 tons, the loading requiring less than three hours, and the unloading ten hours. Now the mines cover more land than was familiar to even the "timber-cruiser" when the Jackson mine was opened. The annual shipments have grown from a few thousand to 10,000,000 tons, and the ability of the mines to produce is not taxed. Lake-freights are at one-eighth, and rail-freights at one-quarter, of early prices.

The deeper waterways, commodious locks and large ships of to-day are in a greater measure due to the movement of ore than to any other cause. Great savings have been made in each detail of each department. The only item of cost that has not been lessened, I am glad to say, is the money paid to men. Wages, measured by what they will buy, are as high

now as at any time past, unless it was a time of feverish activity.

As illustrating the cheapening that has been effected, ore is to-day sold at Lake Erie ports for one dollar per ton less than in early times it actually cost to mine it; and this is done without loss, and further economies may be expected. Young men remember sales of ore in Pittsburgh at $18 per ton, or about the present price of rails.

The first mines were opened in Marquette county and produced what is commonly known as "hard ore." Previous to 1873, "soft ore," or hematite, was not popular, but the buyer was often obliged to take some, in order to secure the needed quantity of hard ore. In those good old days, the ore-buyers came to the sellers as penitents approach a shrine, but I am told that custom in this respect has changed somewhat of late.

Beginning with a few grades, the Lake Superior mines now offer ores ranging from the purest and best to those that will not bear transportation; and within these limits can be found the varieties needed for general and special products, in quantity equal to any demand.

The men who were early in charge of the mines had some knowledge of mining and quarrying. Many of their methods are now out of date; but experience, daily observation and study, technical schools and the mechanical ability common among Americans, have enabled those in charge to meet all difficulties and carry the industry to a degree of perfection that is complimented by visitors from abroad.

In the discovery of new fields, the miner of iron-ore, like the miner of more precious metals, has led the way; the explanation of the geology being left to the student.

As viewed from this distance, the early methods of mining were very crude. Horses furnished the power; and they were worked in all possible places. The little machinery then in use was poor and often but ill-adapted to the work. The first hoisting-plant of large size, economical design and capacity equal to future demands was set up at the Lake Superior mine in 1880. Now single, compound, triple and condensing-engines of large power are found throughout the iron-ranges for hoisting, pumping and compressing. The striking-hammer has been supplanted by the power-drill, and

electric motors and rope-haulage are common above and below ground. Underhand, back- and breast-stoping, square-sets, caving-sets, saddle-backs and milling and filling are common. Steam-shovels and locomotives have taken the place of hand-shovels and carts. Every known system has been or is in use. Many mines have presented difficulties that, in the early days, would have been considered insurmountable; but each has been overcome. To one closely identified with this industry for nearly thirty years, no single change has seemed important, and only by looking backward can one measure the advance. The early operators had little knowledge of ore-deposits, their position in relation to associated rocks, their dip, pitch, strike, depth and many other facts, an understanding of which is necessary to continued success. This knowledge has been acquired; and with it have come right courage, true economy instead of parsimony, the general development of ore-supplies far beyond immediate needs, and a thoughtful preparation for the future.

Nitroglycerine and power-drills have done more to reduce cost than any other two improvements. In 1871 a drift was driven at the Cleveland mine at a cost of $100 per foot. Now the cost would not exceed $16. In one case men were harnessed to a derrick-sweep. Heavy rains were nearly always followed by a few days of idleness and the pumps were expected to freeze every cold night.

In speaking of mining methods, I have mentioned the "caving-system," which several claim to have invented or introduced. The first ore taken out by this method was mined at the Cleveland hematite mine in 1883. The ore was very soft, with treacherous walls; and Mr. G. W. Wallace, the superintendent, tried what some of his men said was successful under like conditions in England. It proved satisfactory, and is now known as the "caving-system." Mr. John Pengilly was sent to the Cleveland hematite mine to learn the method, so that he might use it at the Brotherton mine.

While to the miner the quantity and quality, and not the origin, of the ore have been important, he has still acquired a thorough knowledge of local geology.

A look at the past may tickle our vanity, but our interests are in the future. Can we maintain the present output? Can we increase it? For how many years can buyers depend upon

Lake Superior for their supply? The discovery and quick development of the Mesabi profoundly affected the iron and steel industry of this country; and the effect may extend beyond our shores. The opening of the Mesabi does not mean idleness on other ranges, but does mean for them greater economies and smaller profits. The new range competes with itself as well as with others. But each era of depression brings a reward in the better methods which are forced upon us, and which, continued into days of prosperity, increase the volume of business and to that extent the profits. With the low prices that are here to remain, it is more than ever difficult to estimate the annual consumption. Speaking of the several ranges as one, the acreage of known ore is but a small part of the area that may contain ore. Discoveries are still being made in the Marquette district, the exhaustion of which was expected twenty years ago. And I may say, in passing, that the known ore under the bed of Lake Angeline is four times as great as was all the known ore in all of the mines of that county at any time prior to 1887. The past few years have not been such as encourage prospecting; but given favorable conditions, we may confidently depend upon important discoveries. For forty years this region has met all demands; its resources are to-day, as compared with demand, greater than at any time past; and I believe that those who are now active will have stepped aside and will perhaps have been forgotten before this region will fail as an adequate source of supply.

Methods of Iron Mining in Northern Minnesota.

BY PROF. F. W. DENTON, SCHOOL OF MINES, UNIVERSITY OF MINNESOTA, MINNEAPOLIS, MINN.

(Chicago Meeting, February, 1897.)

MUCH has been written about the possibilities of the Vermilion and Mesabi ranges of northern Minnesota as producers of large quantities of high-grade iron-ore. The Mesabi range in particular has attracted considerable attention on account of the proximity of the ore-deposits to the surface, which has been as-

sumed by many to insure easy and cheap mining. It is thought, therefore, that a description of the methods of mining actually employed may be of interest to members of the Institute.

LOCATION OF THE MINES.

The accompanying map, Fig. 1, taken from Vol. III. of the *Proceedings* of the Lake Superior Mining Institute, shows the positions of the two ranges and the railroads. The two ranges are separated by a line of granite hills running approximately east and west. The geology of neither range is very well understood. A description of the rocks and ore-bodies occurring around Soudan has been previously published in our *Transactions.** The deposits on the Mesabi have also been described.†

The majority of the mines are controlled by the Minnesota Iron Co. and the Lake Superior Consolidated Mines. The Minnesota Iron Co. controls the Duluth and Iron Range railroad, which carries its ore to Two Harbors on Lake Superior, where it is shipped to the lower lake-ports in steamers owned by the company. The railroad-shops and the chemical laboratory are also at Two Harbors, which is practically a company-town. The Lake Superior Consolidated Mines controls the Duluth, Missabe and Northern railroad, and ships its ore from Duluth in its own steamers.

MINES ON THE VERMILION RANGE.

The Minnesota Mine.—At Soudan, the western terminus of the developed portion of the Vermilion range, is the Minnesota mine of the Minnesota Iron Co. From this mine 448,707 tons were produced in 1896; the greater part of which, however, was not Bessemer ore. The ore occurs in lenses, dipping north about 70° and varying in thickness from a few feet to over one hundred. The ore itself is extremely hard and strong, while the green schist ("soap-rock") walls are soft and weak. The method of mining, which may be called longitudinal overhand stoping with filling, has been very thoroughly described by Mr. D. H. Bacon,‡ and I will only supplement his paper.

* "The Geological Structure of the Western Part of the Vermilion Range, Minn.," by Messrs. Smyth and Finlay, *Trans.*, xxv., 595.

† "The Mesabi Iron Range," by H. V. Winchell, *Trans.*, xxi., 644.

‡ "System of Filling at the Mines of the Minnesota Iron Co.," *Trans.*, xxi., 299.

Figs. 2 and 3 are underground views from this mine. Fig. 2, taken on top of the "fill," shows a chute tapping a raise in the foot-wall from which rock is obtained for filling. As the

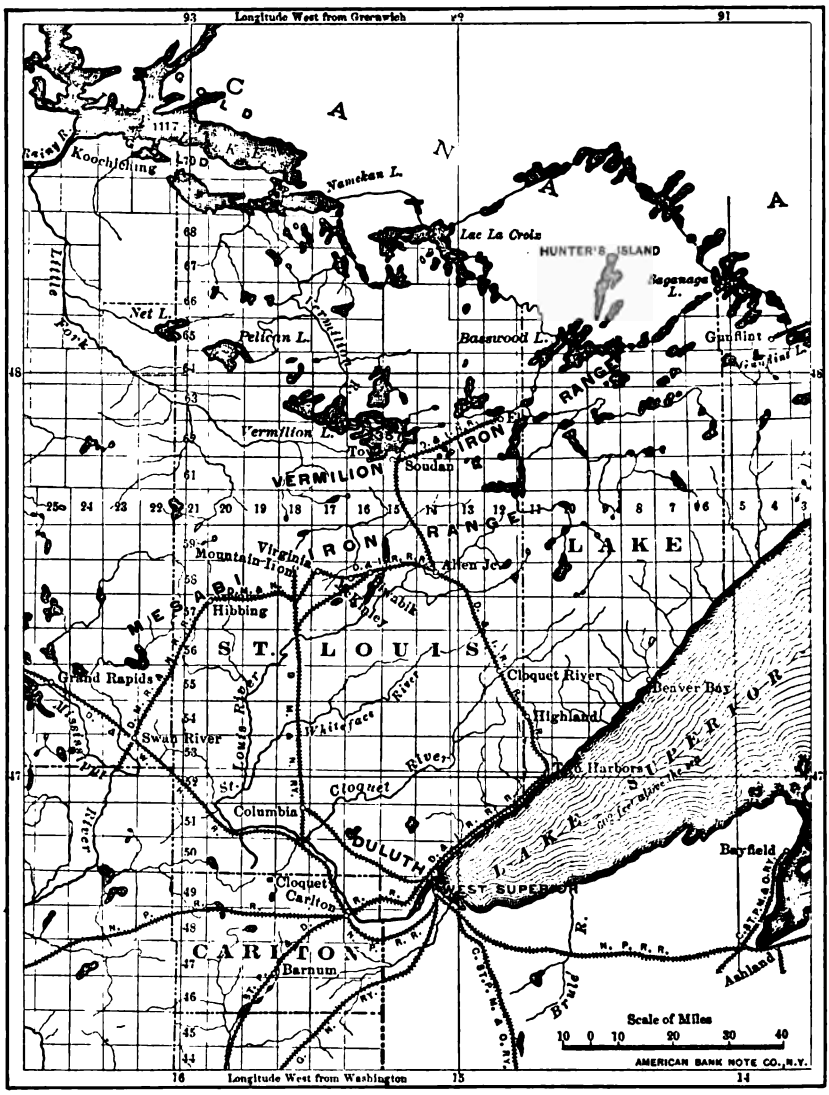


FIG. 1.—Map of the Iron Ranges of Northern Minnesota.

back of ore is blasted away and the level of the fill rises, the rock-raise is tapped higher up. The side of the raise, at first formed by the ore, is maintained, after the ore is removed, by

timbers; or else the rock-raise is replaced by a timber chute carried up with the fill, either of which arrangements will carry the rock-raise through the fill to the level below, where it may be used in mining a lower block of ore.

The car used for distributing the fill is also shown. The box of the car can be swung in a horizontal plane so as to dump either to the right or left side of the track as well as over the end, which permits of making a wide fill for each position of the track.

The diamond-drill, shown in Fig. 2, is used for drilling holes in back-stoping. A Sullivan "E" drill, run by compressed air, is used. The drill is mounted on cribbing and braced as shown. The holes vary in depth from 20 to 33 feet with an average of about 25 feet. About 12 feet are drilled in 10 hours. The holes are first shaken by filling them with dynamite and firing. In a stope 40 feet wide, 2 holes, 24 to 30 ft. deep, would be put in, one pointing a little to the hanging-wall and the other a little to the foot-wall, and charged with 30 to 50 pounds of 50-per-cent. dynamite. These holes would dislodge between 500 and 1,000 tons of ore.

The cost of the drilling alone exceeds the cost of drilling with machine percussion-drills; but the product per foot of hole is much greater with the diamond-drills, making the cost per ton less.

Fig. 3 shows the main-level timber being put in, preparatory to making the first fill, and needs no special explanation. The chutes are now carried closer than formerly, being placed about 30 feet apart, which lessens the labor of getting the ore into the chutes. By experience it has been found that 100 feet vertical is the economical limit for carrying up timber chutes, as the cost of repairing and replacing timbers that are cut out by the falling ore, and the delays due to the chutes becoming clogged, are more expensive than to put in another main level, and a new series of chutes.

The floor-pillars of ore, which have to be left under the old fill above any given stope, are mined by attacking them from drifts in the foot-wall, run at the same level as the top of the old fill under the pillar to be mined. From these foot-wall drifts, cross-cuts are put into the ore-pillar on a level with the top of

the old fill; and the ore which has caved on the fill (or if not, it is made to cave) is taken out with drift-sets.

I know of no deposit which is so well adapted by nature to the system of mining employed as that at Soudan, and no place where every natural advantage is utilized more effectively.

Since the lenses of ore pitch to the west, the bottom levels (the 9th to the 12th) are practically all west of the shafts, necessitating a tram of 500 to 1,000 feet. The conditions did not warrant the sinking of a new shaft; so mules were introduced to lessen the cost of tramming, and they have proved very satisfactory.

The pumping-plant of No 8 (the deepest) shaft, is of a type not used by any other company in the Lake Superior region, and not in general use elsewhere, to my knowledge.

Fig. 4 shows as much of the pump as could be included in one view. There are two of these pumps set up alongside each other but not connected; one being practically a spare pump, as the amount of water is seldom more than one pump can handle.

The same kind of pump is used at the other mines of the Minnesota Co., and data for all of them are included in the following table:

	Soudan mines.	Auburn mines.	Fayal mines.
Diam. of steam cylinders,	12 in.	12 in.	11 in.
Diam. of plungers,	4 in.	6¼ in.	5 in.
Length of stroke,	24 in.	24 in.	16 in.
Diam. of fly-wheels,	8 ft.	8 ft.	4 ft.
Weight of fly-wheels,	4 tons.	4 tons.	1900 lbs.
Steam-pressure at pumps,	90 lbs.	95 lbs.	90 lbs.
Diam. of column-pipes,	8 in.	8 in.	6 in.
Vertical lift,	817 ft.	about 150 ft.	about 150 ft.

Each pump is supplied with a Meyer's cut-off which may be adjusted by means of the hand-wheel shown in the view. Throttling-governors are also attached to the pumps to prevent the engines running away, should the load be suddenly taken off—as happened once when the column-pipe was emptied by the breaking of a valve-cover on another pump which was connected temporarily. The governors are set for 90 revolutions per minute. The cut-off varies with the speed, and is designed to be at one-fifth stroke, with a piston speed of 100 ft. per

minute. The speed at which the pumps can be run varies with the lift. No air-chambers were used at first; but very recently air-chambers, made of 6-inch extra-heavy pipe, have been placed over the discharge-valves. By means of a check-valve and shift-

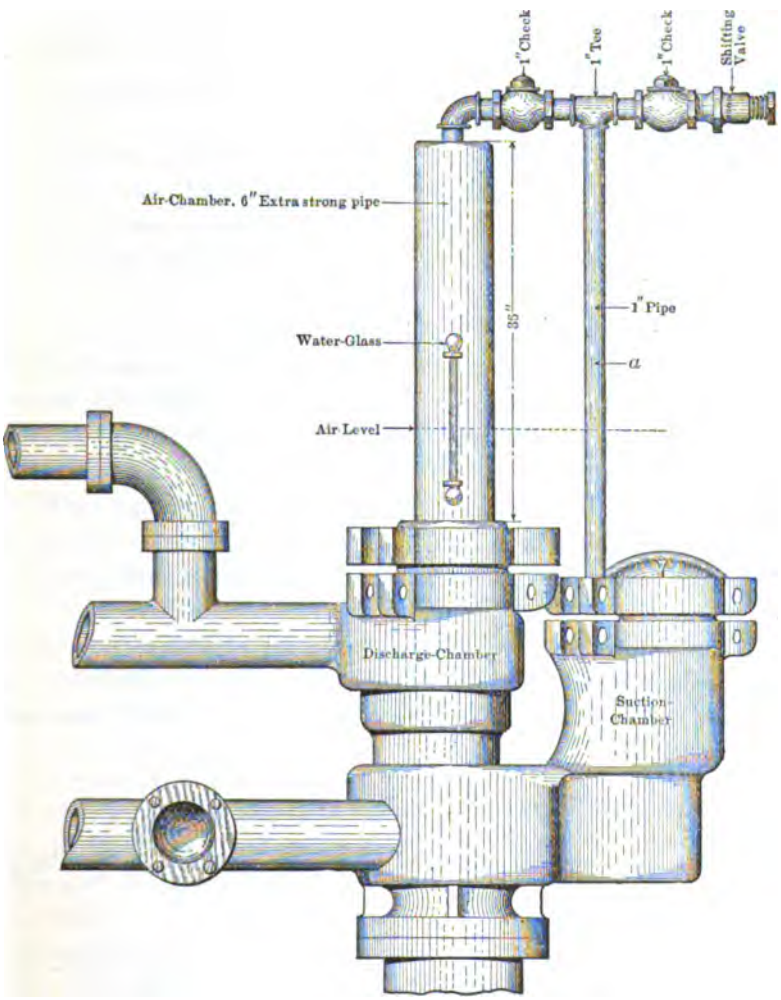


FIG. 5.—Device for Supplying Air to Air-Chambers.

ing air-inlet valve, connected by pipes to the air-chambers and suction-valve chambers, the air-chambers may be kept supplied with a constant volume of air, which otherwise would be continually carried off by the water under pressure.

Fig. 5 shows the arrangements of the air-chambers and piping, the operation of which is as follows: While the water-cylinder is filling, the column of water in the pipe, *a*, is drawn downwards by suction, the amount drawn off depending upon the amount of air admitted by the shifting-valve; on the return-stroke of the plunger of the pump, the water in the pipe, *a*, rises and forces air through the check-valve into the air-chamber. The quantity of air is regulated by the shifting-valve, enough being admitted to keep the proper amount of air in the air-chamber, as shown by the water-glass. These air-chambers have permitted an increase in the plunger-speed from 100 feet per minute to 360 feet, without perceptible water-hammer. Without the air-chambers water-hammering began at 100 feet plunger-speed.

The water-ends of the pump are of the Prescott type.

The foot of the column-pipe is supplied with gate- and check-valves as shown. Indicator diagrams, taken from the steam-cylinders, are very satisfactory. No accurate duty-test has yet been made upon these pumps; so the duty is not known. Should the water increase sufficiently, the two pumps could be coupled together, in which case the air-chambers could be dispensed with and the efficiency of the plant would probably be increased.

As at other hard-ore mines, plants for crushing the ore to sizes suited for furnace-use have recently been established at Soudan. There are two crusher-plants, of identical size and arrangements.

Data Concerning Crushers.

Each plant contains three 28 by 30 in. crushers of the Blake pattern.

Throw of eccentric,	2 in.
Movement of bottom of jaw,	1¼ in.
Weight of pitman,	5 tons.
Weight of jaw,	7 tons.
Weight of crusher,	65 to 70 tons.
Speed of crusher,	85 rev. per m.

Each plant is driven by a Reynolds-Corliss engine and rope-drive as follows:

Cylinder,	14 in. diam. by 36 in. stroke.
Speed,	90 rev. per m.
Initial steam-pressure,	95 lbs.
Diam. of fly-wheel,	10 ft.
Weight of fly-wheel,	8 and 10 tons.
Distance between centers of engine-shaft and counter-shaft for rope drive,	75 ft.
Diam. of manilla ropes,	1½ in.
Number of ropes,	6
Number usually in use,	4

The features of these crushers of special interest are the lining-plates, the size, and the extremely hard work they have to do. The Calumet and Hecla copper mine has crushers of larger size (24 by 36 in.); but the ore at Soudan is much harder to crush. At first, trouble was experienced from the stalling of the crushers; but the power of the crusher was made a maximum by flattening the angle between the toggles as much as possible, and increasing the throw of the eccentric to get the necessary movement of the jaw. The lining-plates also broke; and even when they did not break, the lower parts would be worn out when the upper portions were still serviceable. Mr. H. J. Wessinger, the master-mechanic, finally designed and adopted the style of plates now in use.

The side-liners consist of three pieces of manganese-steel, riveted to a tank-steel back. The jaw and head-plates consist of four pieces, held together by iron links, of 1½ in. square section, shrunk on. As the bottom-plate is used up, those above it are moved down, and a new piece is attached to the top, where it will fit, as there is no appreciable wear at the top. In this way the metal in the plates is entirely used up.

All kinds of hard metal have been tried for the wearing-faces of linings: chilled iron, tank-steel, armor-plate (not Harveyized), cast-steel and manganese-steel. An experimental manganese-steel plate is now being tried, in which the face of the plate has large horizontal corrugations. The manganese-steel is liked as wearing well; but as it cannot be touched with a tool it is difficult to make the blocks fit.

In a communication to the Lake Superior Mining Institute* the cost of crushing is given as follows:

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* *Proceedings of the Lake Superior Mining Institute*, vol. iii., 1896.

"During three months ending January 1st last (1895), 110,477 tons of ore passed through the crushers, costing for supplies $5,025, or 4.54 cents per ton, and, grouping the other accounts, $3,718.47, or 3.36 cents per ton; total, $8,743.47, or 7.9 cents per ton. We have no means of determining what percentage of the ore needs to be crushed, but assuming it to be 60 per cent., the cost was 13.19 cents per ton."

The ore, as it is dumped from the cars, runs over grizzlies directly into the opening of the crushers; so there is no intermediate crusher-floor, with the extra handling which such a floor entails. When possible, the cars are carried from the shafts to the crushers by self-acting inclines, each car being pushed by a "bull-dog," which runs on one of the main rails and a special third rail laid between the main rails. When the bottom of the incline is reached, the bull-dog runs into a depression in the middle of the track, and the car is carried by its impetus on to the crushers.*

The Chandler Mine.—This mine, located at the eastern end of the developed portion of the Vermilion range, is famous for the high and uniform quality of its ore. Last year it shipped 471,545 tons, of which about 87 per cent. was of "Chandler" grade, containing 65.65 per cent. of iron and 0.048 per cent. of phosphorus (less than the Bessemer limit), while the remaining 13 per cent., of "Long Lake" grade, contained 60.64 per cent. of iron, and was also Bessemer ore.

The deposit is supposed to have been originally massive hematite, which was subsequently broken up by natural forces, such as folding. This view is supported by the form of the ore-body, and by the fact that the individual pieces of the ore are very hard hematite. There is a peculiar layer of altered schist, about thirty feet thick, between the ore and the unaltered green schist. This layer is locally termed "paint-rock."

Fig. 6 shows the extent of the ore-body at the eighth level, and Fig. 7 is a vertical section. It will be seen that the ore lies in a fold or basin, dipping slightly to the north, and is covered by a jasper capping. The basin pitches to the east at a small angle. At Soudan, the pitch is to the west.

* This arrangement, being substantially the same as that described in the paper of Mr. Axel Sahlin, on "The Handling of Material at the Blast-Furnace," need not be further described. See pages 14 and 29 of the present volume.

Fig. 6 shows the positions of the shafts that are now used. Shaft No. 1 is located at the western extremity of the deposit, where the ore was first discovered and opened up, but the

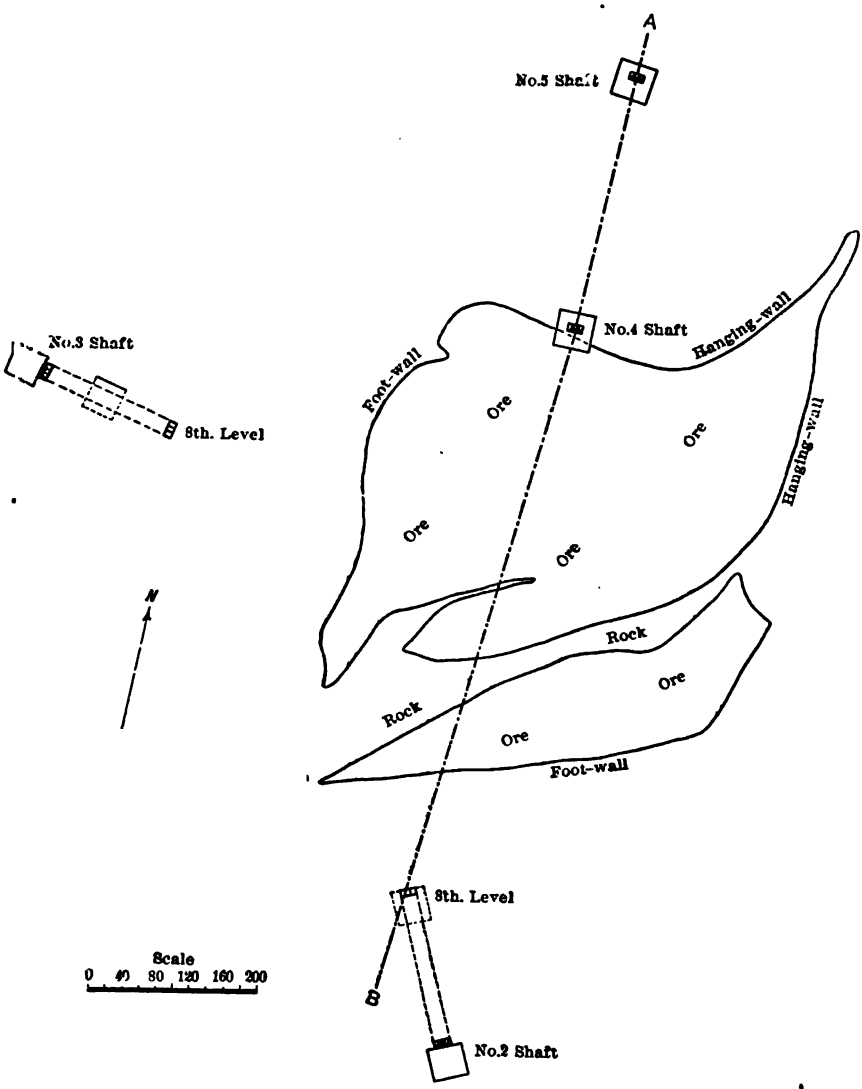


FIG. 6.—Outline of the Chandler Ore-body at the Eighth Level.

shaft has been worked out. The bulk of the ore now comes from below the sixth level. As is indicated in Fig. 7, there has been considerable trouble in the past from the wrecking of the tops of the shafts, due to the cave extending further than

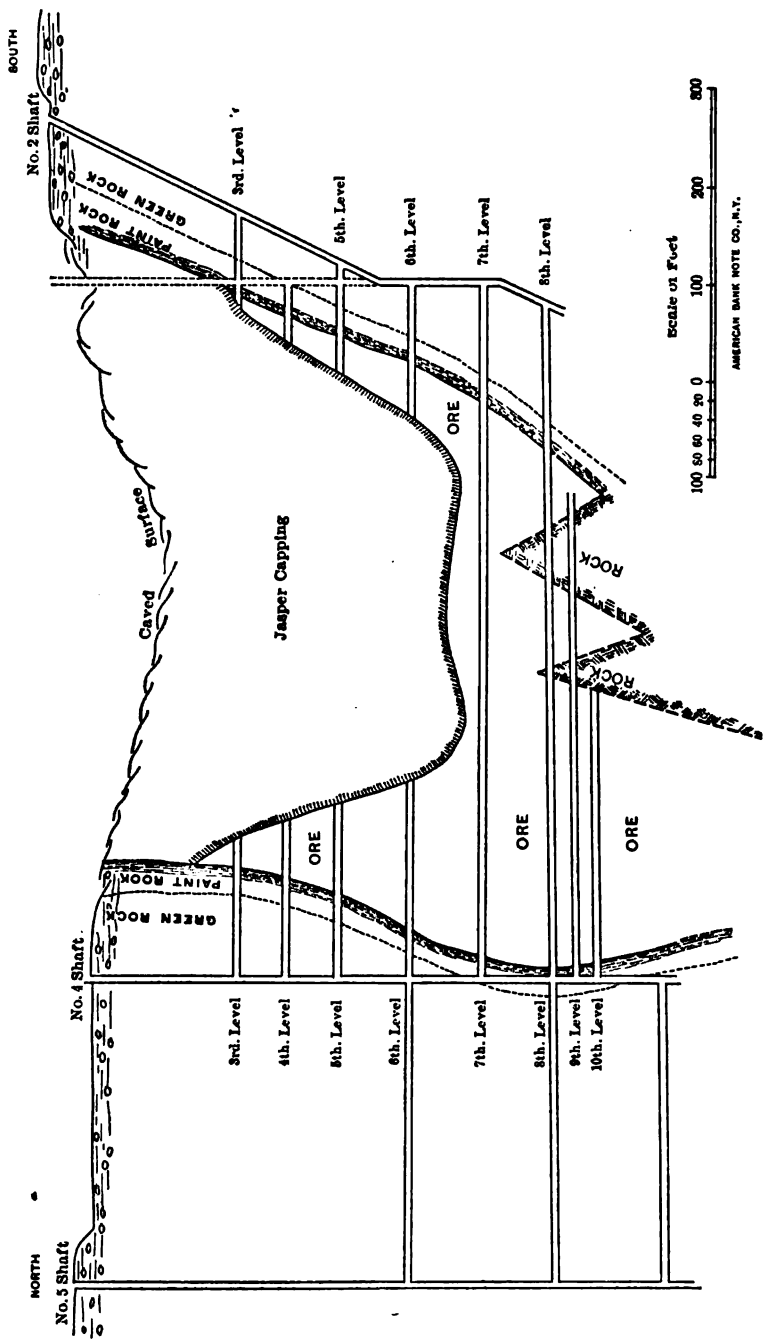


FIG. 7.—Vertical Section of the Chandler Ore-body along A–B of Fig. 6.

anticipated—a difficulty experienced by every mine on Lake Superior that has employed caving methods. Shafts Nos. 2 and 3 were originally vertical, but were subsequently inclined, to follow the dip of the walls. When the tops of these shafts were lost by their sliding into the pit, they were recovered by sinking inclined shafts further back to meet the uninjured portions of the original shafts. Shaft No. 4 is vertical all the way; but by reason of the turn which the ore took to the north, a large block of ore has to be left as a pillar. To get at the ore thus locked up, and to replace No. 4, which must soon be lost, No. 5 was sunk, and will eventually become the main hoisting-shaft.

Fig. 8 shows shaft No. 3 and the edge of the cave. The same style of head-frame and pocket is used for all the inclined shafts. It is an excellent design.

The method of mining employed is known as the sub-drifting method. Fig. 9 is a vertical section through shafts Nos. 4 and 5, and shows the way the ore has been blocked out. Down to the eighth level the method of mining is as follows: Main levels are driven 75 feet apart and generally there are two main drifts at the bottom of each block, running approximately parallel on opposite sides of the block of ore. From these main drifts raises are put up at intervals of about 50 feet, and from these raises four series of sub-drifts are run. The sets in the main drifts are made of 9-ft. caps and 7-ft. legs, and those in the "subs" of 6-ft. caps and 6-ft. legs. This leaves about 8 feet of ore between the sub-levels. The sets are placed 3 to 4 ft. apart. As the raises are put up, sets are placed to start the first sub-drifts; but these drifts are not run at once, but are omitted to strengthen the main drifts until the 4th, 3d and 2d subs have been worked out. When the sub-drifts are completed, the block of ore between any two levels is honeycombed with drifts with vertical intervals of eight feet of ore. When mining above has been completed, the removal of the ore-pillars on the top subs begins. The pillars are sliced away, the back is caved, and the caved ore is removed in wheelbarrows to the chutes leading to the main level below. The chutes are 4 feet square and lined with 2-inch plank placed on edge. When the sand or overlying timber appears, a new slice is taken off the pillar and the back of ore is caved as before,

until finally all of the sub-drifts have been worked out, when the operation of caving is continued in the block below, which, in the meantime, will have been honey-combed by the first or preparatory sub-drifts.

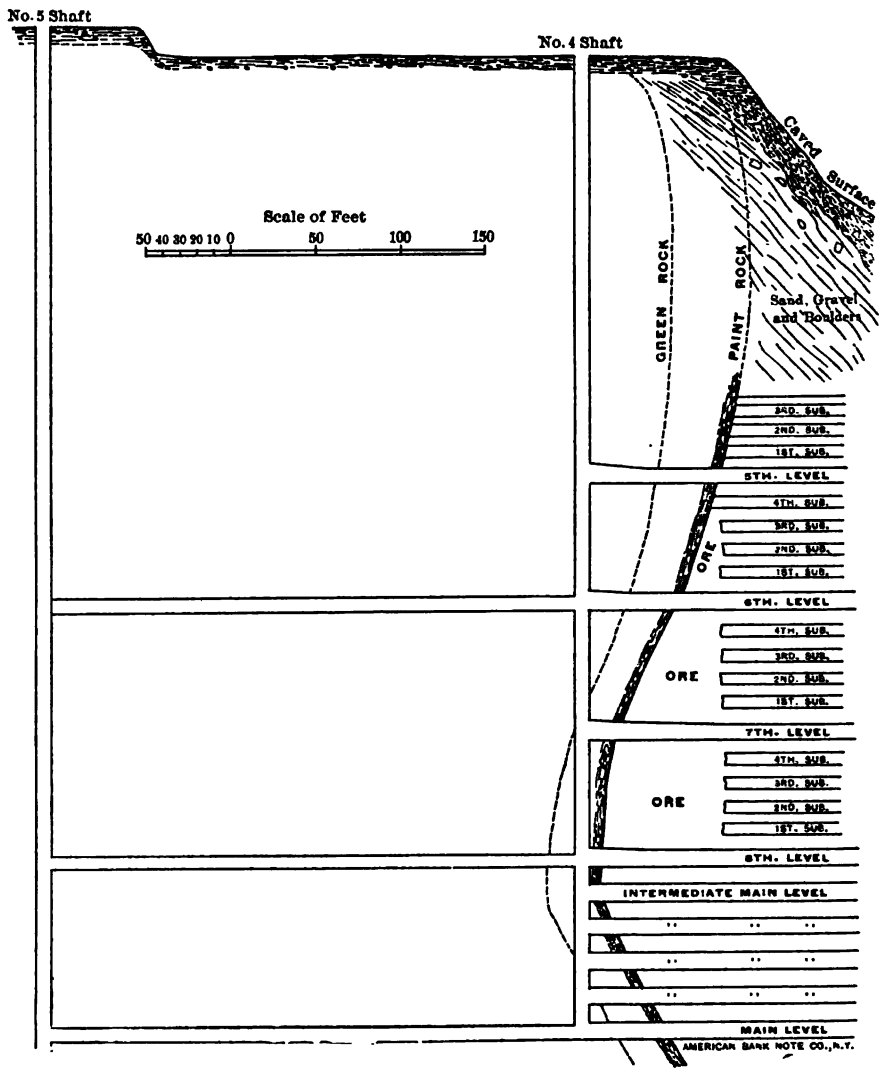


FIG. 9.—Vertical Section through Shafts Nos. 4 and 5, Chandler Mine.

Below the eighth level the method of mining has been modified. What are called intermediate main drifts are driven through the ore at intervals of 20 feet instead of 75 feet, and no sub-drifts are used. The intermediate main drifts are of

the regular size, 9-ft. caps and 7-ft. legs, which leaves about 10 feet of ore to be caved, instead of 7 to 8 as before. Stations are made at the shaft for each intermediate main level. Under this modified system, the removal of each 20-ft. block will be done as before, but the putting up of raises will be saved; and it is intended to use cars, and thus do away with wheelbarrows as far as possible.

Fig. 10 shows a main drift which has begun to cave under the weight caused by the wrecking of the sub-drifts just above.

Owing to the presence of sand and broken "paint-rock" in the caved ground above, the ore obtained by the caving-process is more or less mixed with these impurities, and the percentage of iron in it is lowered from 64 per cent. to 60 per cent. This ore, however, is obtained very cheaply; and it is claimed that this compensates for the injury to its quality. Although broken up, the ore in its undisturbed state is really quite hard. In driving main drifts, machine-drills have to be employed frequently.

Top-lagging is used in the sub- and main drifts, and side-lagging where necessary, but no attempt is made to lag the bottom of the sub-drifts before caving the back.

The Pioneer Mine.—This mine is on what is thought to be the continuation of the Chandler deposit; and, as the pitch is east, the Pioneer has to do all the pumping for both mines, since the water from the Chandler percolates through the intervening ground.

Mines on the Mesabi Range.

On the Mesabi range are found occurrences of ore which, at the time of their discovery, were entirely new to the Lake Superior region. The deposits are beds of great area and relatively small depth, dipping to the south at low angles, and covered with glacial drift, varying in thickness, in the developed mines, from 2 or 3 to 100 ft. One of the largest of these deposits, the Biwabik, has been described in our *Transactions.**

Although generally considered to be a district of open-pit mining, the Mesabi has really only five deposits which are

* "The Biwabik Mine," by H. V. Winchell and J. T. Jones, *Trans.*, xxi., 951.

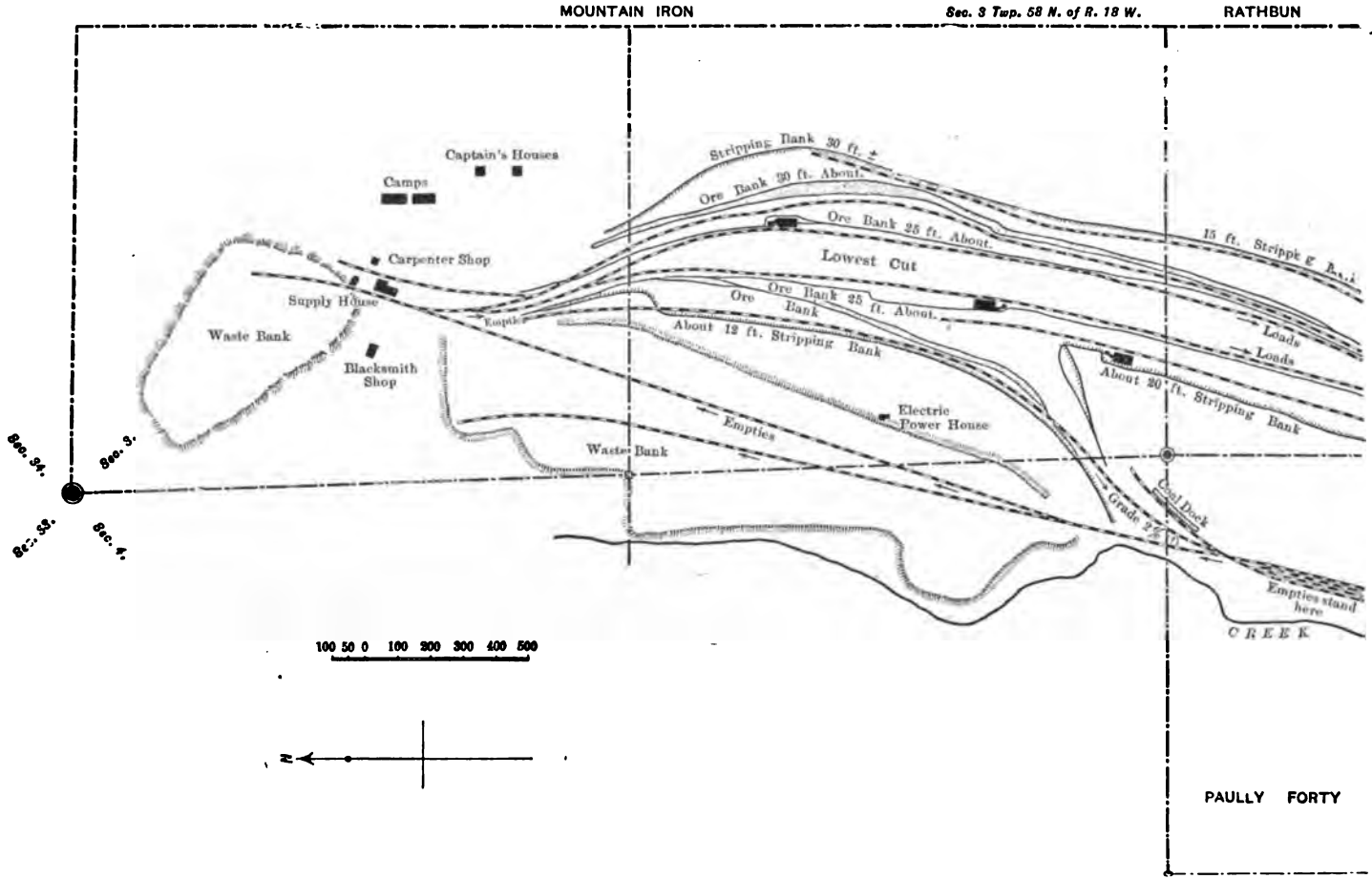


FIG. 11. Plan of the Mountain Iron Mine.

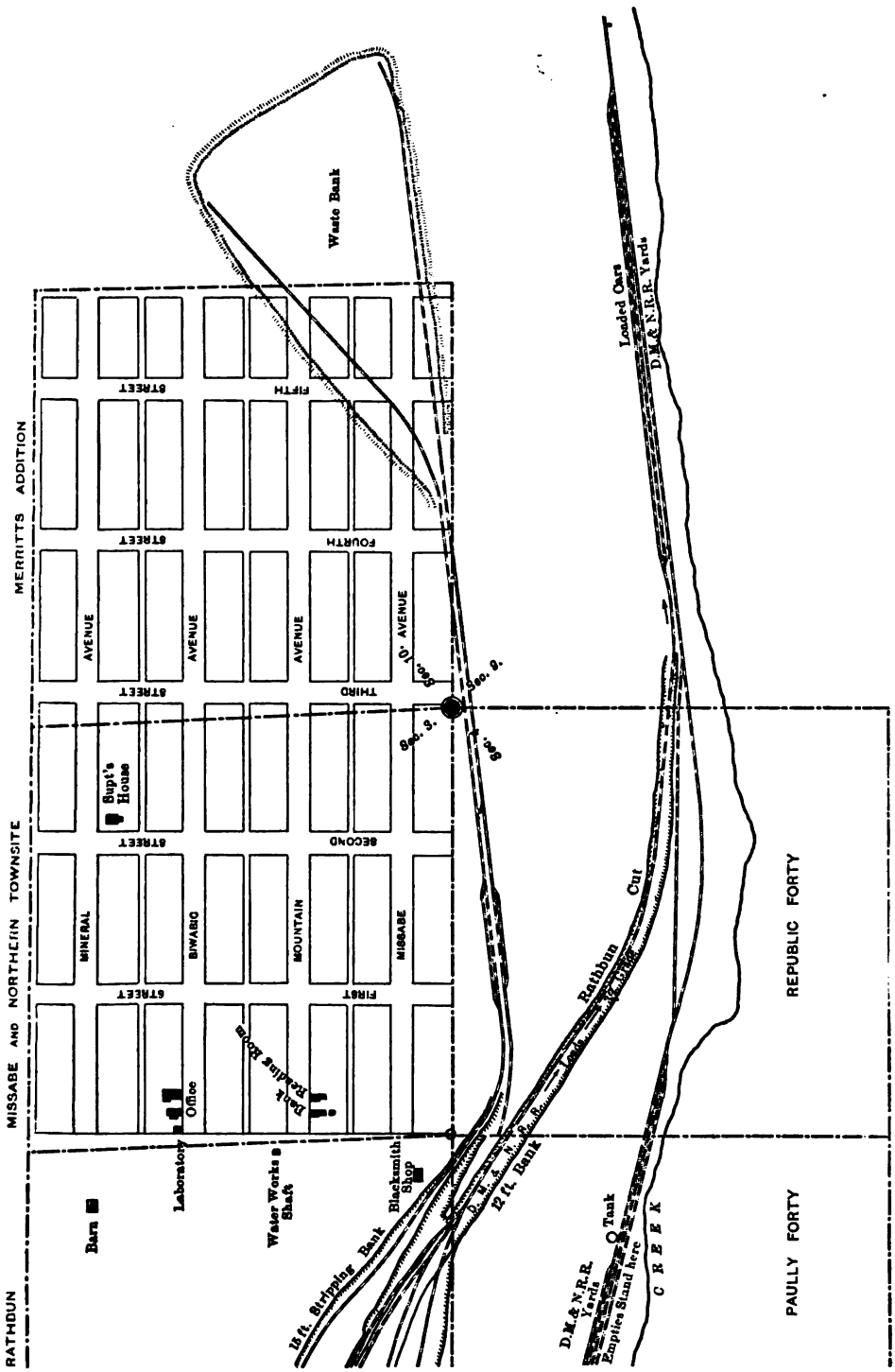


FIG. 11 continued.—Plan of the Mountain Iron Mine.

worked strictly by open pits. There are about twelve purely underground mines, and three or four that use a combination of underground and open-pit methods. The output for the past year, however, gives the lead to the open-pit mines, which have furnished about 1½ million tons, while the product of the underground mines was about one million, and that of the others about half a million.

The Mountain Iron Mine.—This is the ideal open-pit mine of the Mesabi range. The topography of the deposit is especially adapted to mining with steam-shovels. Fig. 11 is a plan of the mine, and shows the cuts into the ore, the number of benches and the track-system.

The first work was done at the extreme north end, and later the other two cuts were put in. The map needs little explanation. The empty cars are collected in the central yard and are carried, in trains of 12 to 15 cars, to the north end of the pit to approach the shovels from that end, while the loaded cars pass out of the Rathbun cut to the lower yards. The water is handled by natural drainage through a ditch, about 12 feet deep, made at the side of the track in the Rathbun cut. Both outlets for loaded cars have grades opposing the load, as indicated. Positions for three shovels are shown, two loading ore and one stripping. Two locomotives are used for each shovel; one to handle the empty, and one the loaded cars. The only delay, therefore, that necessarily occurs is when the shovel is moved forward. Some great records in steam-shovel work have been made here. The best record with the largest shovel is said to be 171 26-ton cars, or 4,446 tons, in 10 hours; and an average for one month went as high as 133 26-ton cars per day of 10 hours, or 3,458 tons, without the employment of night-shifts. The shovels used were not exceptionally large or of special design.

The ore is first loosened by blasting. A shovel-crew consists of five pit-men, an engineer, a cranesman and a fireman. It is probable that ore can be mined more cheaply here than anywhere else on the Mesabi range, or in the Lake Superior region.

The grades of ore guaranteed last year and the amounts shipped were:

	Iron. Per cent.	Phosphorus. Per cent.	Amount. Tons.
Mountain Iron grade, . . .	64.00	.045	77.698
Helmer grade, . . .	64.00	.055	34.313
Tubal grade, ...	62.00	.065	20.990
Total, ...			133,001

In order to classify the ore into these grades, the banks are carefully sampled from 25 to 50 feet ahead of the shovel, depending upon the nature of the ground, and the results are used to guide in loading the cars. Samples are also taken of every train-load, and of cargoes, if necessary.

Fig. 12 shows a shovel at work at the Biwabik mine in one of the two cuts started according to the elaborate plan laid down in the description of this mine previously referred to. The Biwabik is a magnificent deposit, but has encountered some hard times in its development. Last year 243,565 tons were mined. The cuts are deep, as shown in the view; and the facilities for handling cars are poor.

The Oliver Mine.—This mine, located at Virginia, is also worked by open pits. Fig. 13 shows the latest of the special Vulcan steam-shovels, at work in this mine. The shovel is said to weigh 80 tons, the dipper holding 5 tons. The ore lies in a gently sloping side-hill; and after the stripping had been done, a cut was carried into the ore starting at the foot of the hill. This cut was then widened by alternate stripping and slicing with the steam shovel. The grade of the tracks is now considerably against the loaded cars; and the water-level in the ore has been reached. To drain the ore, a shaft has been sunk, and small drifts have been run in the bottom of the deposit, through which the water is collected and pumped to the surface. A great deal of ore has been taken out of this cut. Last year the output was 808,291 tons.

The Auburn Mine.—As an example of the mines employing underground mining combined with stripping, I have selected the Auburn mine. This method of extraction is locally termed the "milling" system.

Fig. 14 is a plan of the underground openings. Thus far, two levels have been opened out; the first level having about 65 feet of ore over it, and the second about 60 feet. The first level was laid out on what may be called the diagonal plan. A

main drift was run straight out from the bottom of the shaft, and from this the branch drifts were opened at an angle of 45°. All the drifts were timbered. From all of the drifts raises were carried up to the top of the ore, which had been previously uncovered. The amount of stripping done in 1895 is in-

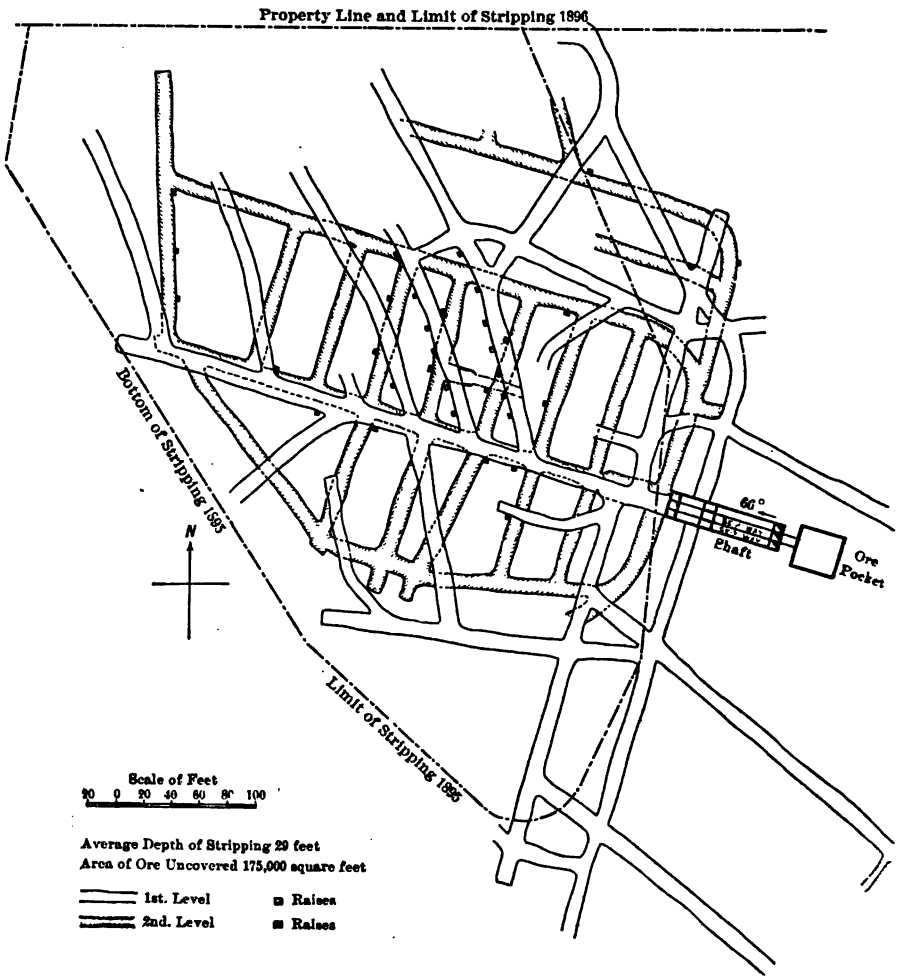


FIG. 14.—Auburn Mine, Minnesota Iron Co., Mesabi Range.

dicated on the map. The stripped area has since been increased, and now reaches nearly to the shaft and extends over the drifts in the northeast corner of the mine. The second level was opened on the rectangular plan, that is, the branch drifts were run at right angles to the main drift from the shaft.

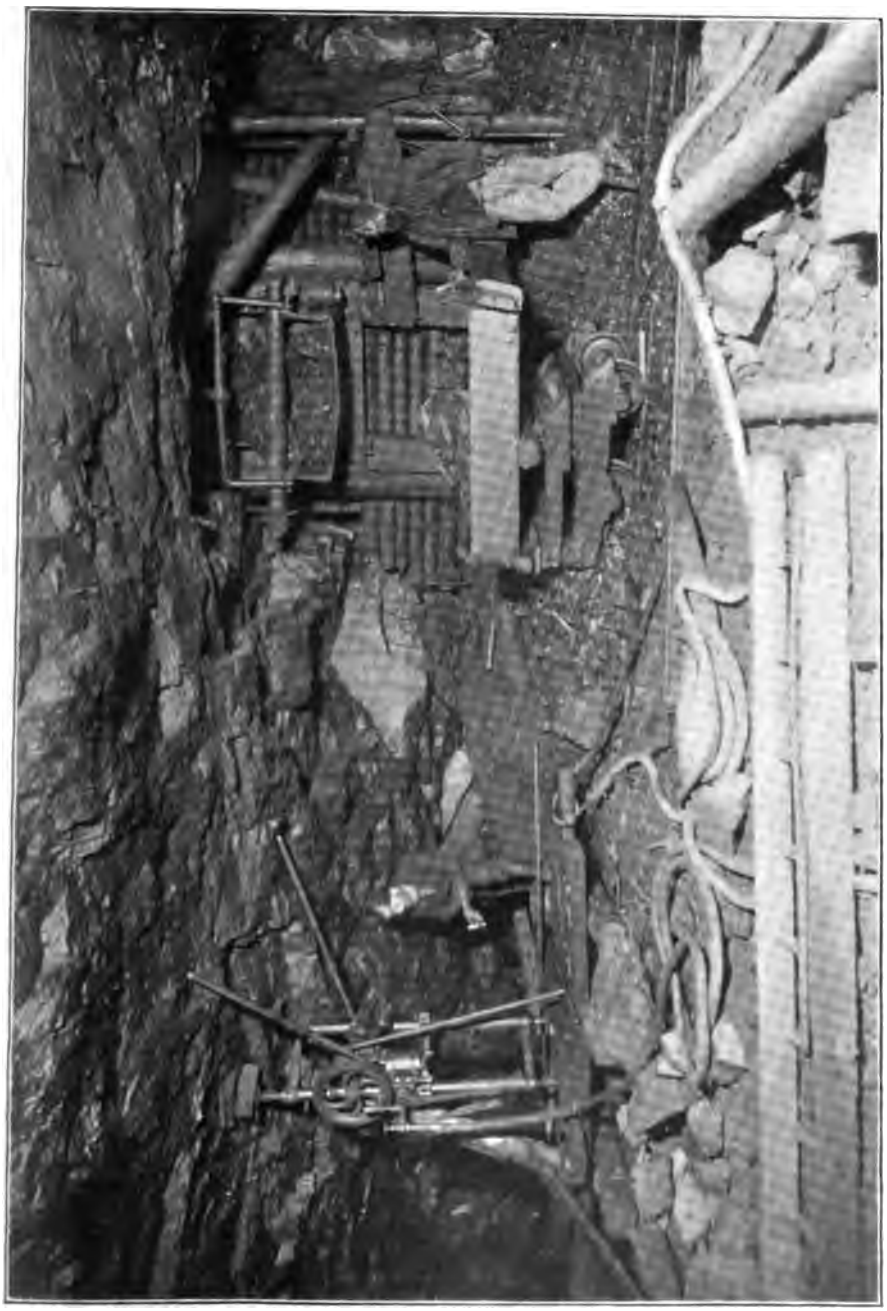


FIG. 2.—View on Top of a Fill in the Minnesota Mine.

FIG. 3.—View of Main Level Timbering, Minnesota Mine.

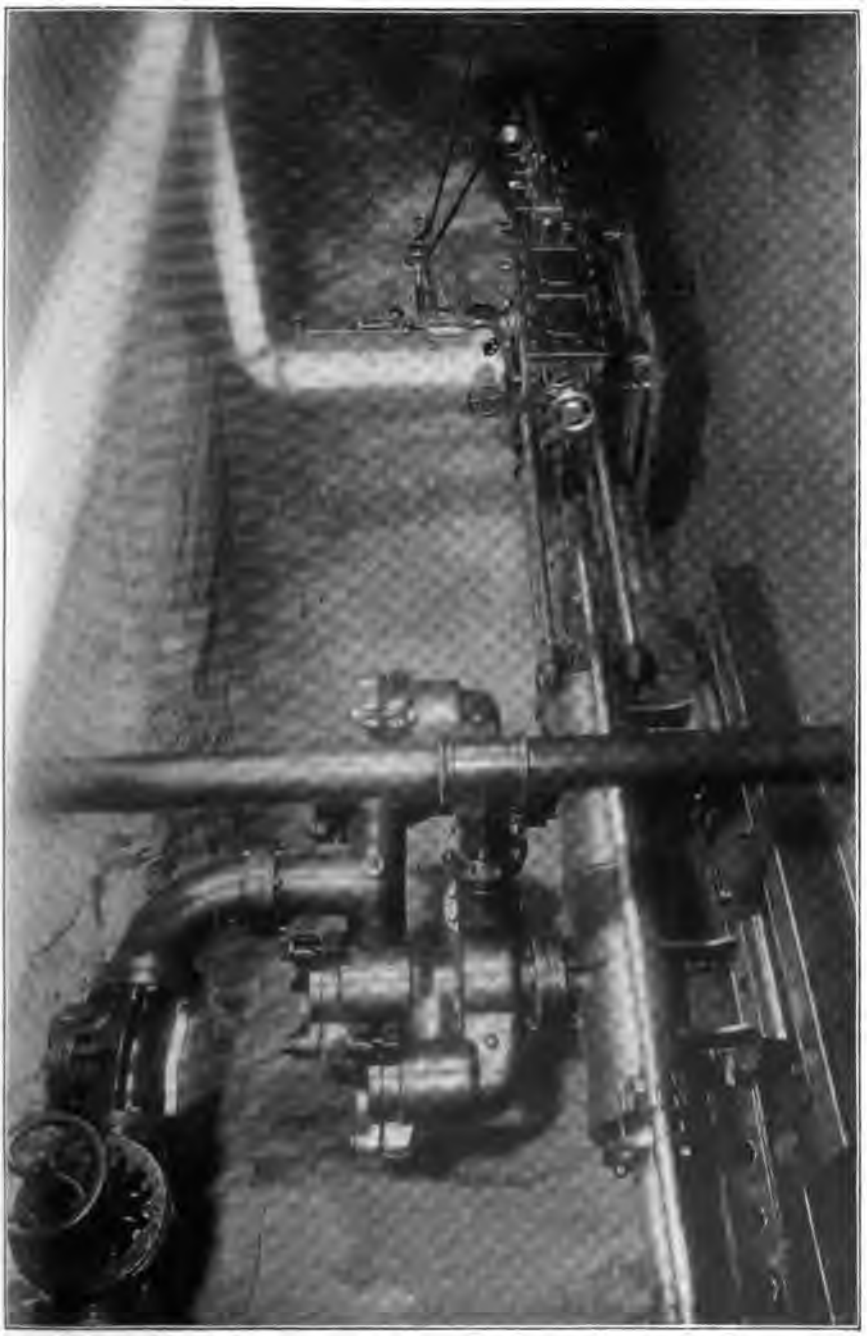


FIG. 4.—Underground View of Fly-Wheel Pump, Minnesota Mine.

FIG. 8.—View Showing No. 3 Shaft and a Portion of the Cave, Chandler Mine.

FIG. 10.—View of a Main Drift which has Begun to Cave, Chandler Mine.

FIG. 12. View of the West Cut, Looking North, Hiawatha Mine.

FIG. 13.—View of an 80-Ton Vulcan Shovel, Oliver Mine.

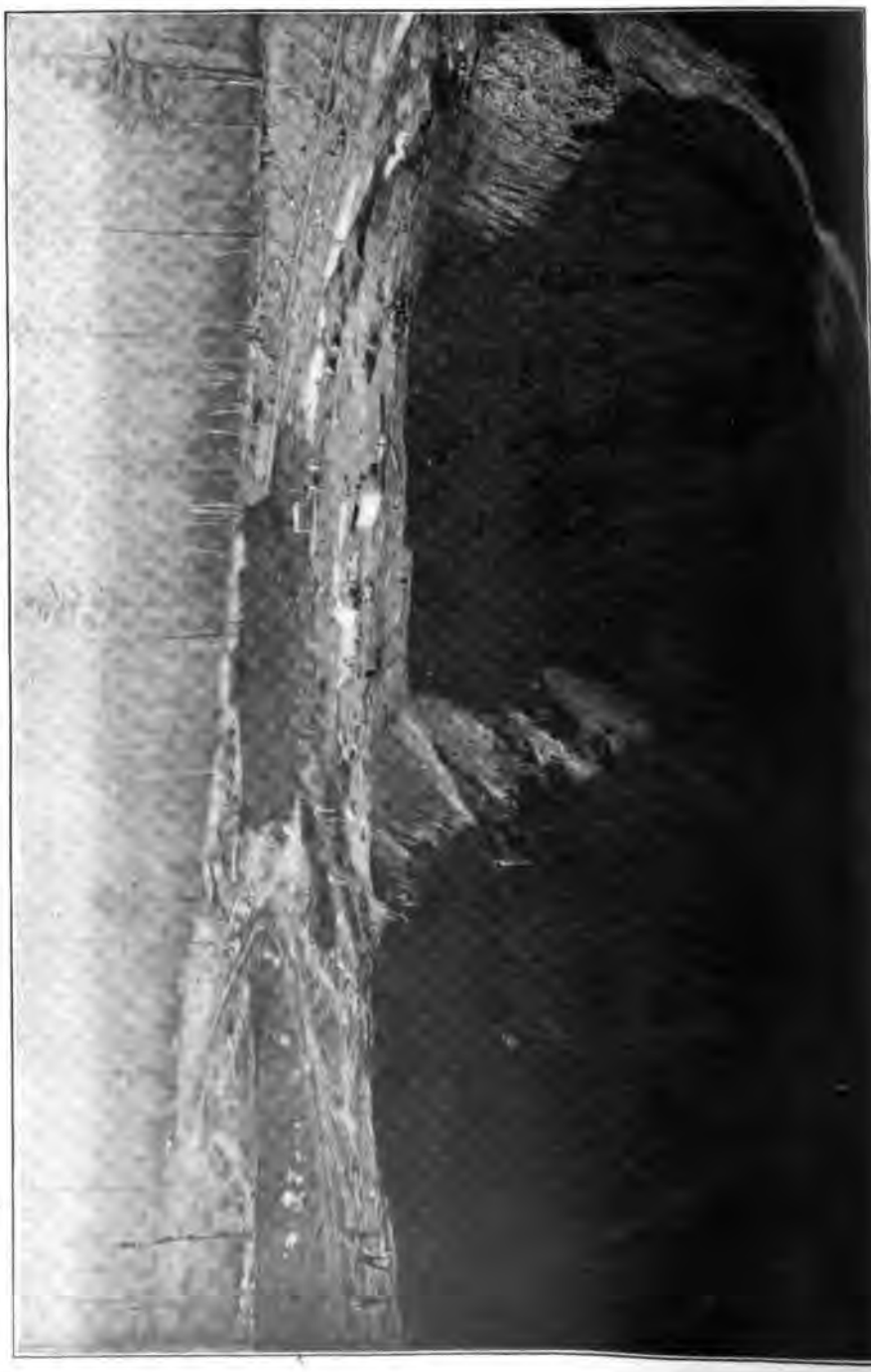


FIG. 15.—View of the Tops of Raises through which Ore is Milled to the Level Below, Auburn Mine.

FIG. 16.—View of the Auburn Mine After the First Level Drifts had been Reached.

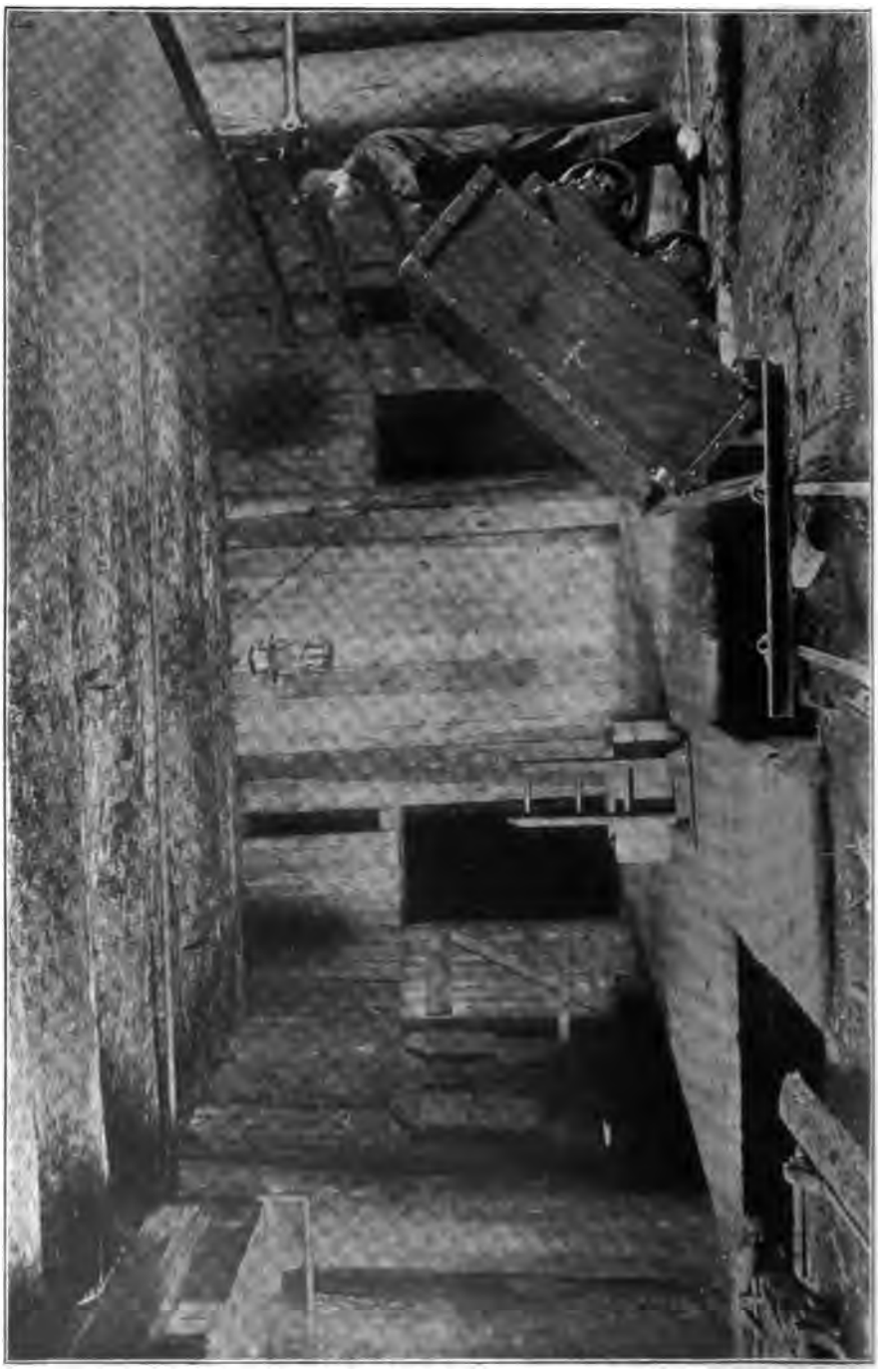


FIG. 22. View in the Fayal Mine, Looking towards the Shaft, and Showing the Openings into the Pockets.

FIG. 23.—View at the Bottom of a Timber Slide, Fayal Mine.

FIG. 25.—One of the Steam Shovels ' by the Mahoning Co.

When the raises from the first level had been completed, the ore was "milled" down through them from the surface, trammed to the shaft and hoisted.

The first raises are placed 40 feet apart; and when these are mined out, a second series of raises is put up half-way between the first to tap the pyramid of ore left between the first. The raises occasionally become "hung up," or clogged, by the ore sticking or becoming jammed in them; but by preventing large lumps from getting into the raises, and not letting the raises become too full, or the ore stand in them too long, no serious difficulty is experienced from this source. By making the raises larger at the bottom than at the top, this difficulty might be still further diminished.

Fig. 15 shows the work of milling in progress at the top of the first raises. Holes are drilled around the sides of the craters by driving down, with sledges, pointed 1¼-in. steel rods, varying in length from 5 to 18 feet. The time required to drive these rods varies with the hardness of the ground and the depth of hole, and may be from ten minutes to an hour. Two to four strikers may be employed. To draw these rods, which often stick very tight, clamps are fastened to the tops and railroad-jacks are placed under the projecting arms of the clamp. When a rod has been drawn the first 6 or 8 inches, it can generally be drawn the rest of the way without much trouble. These holes are first "chambered" with three or four sticks of 50-per-cent. dynamite and then charged with three to ten 25-lb. kegs of black powder.

To prevent men from being carried into the raises by the sliding of the ore on the sides of the craters, ropes are kept within their reach. Owing to the neglect of this precaution on the part of the men, accidents of this nature have happened. Men carried into the craters have been rescued by placing around them barrels, pieces of smoke-stack or any tube large enough to keep the ore from smothering them. With this protection the men have time to work themselves loose, and can be pulled out.

Fig. 16 shows the condition of affairs after the first level has been reached.

By constantly sampling the ore, it is graded into two classes, carrying respectively 64.95 per cent. of iron and 0.049 per cent.

of phosphorus, and 62.5 per cent. of iron and 0.065 per cent. of phosphorus. A total of 325,000 tons was mined the first year (1895), and 131,000 tons in 1896.

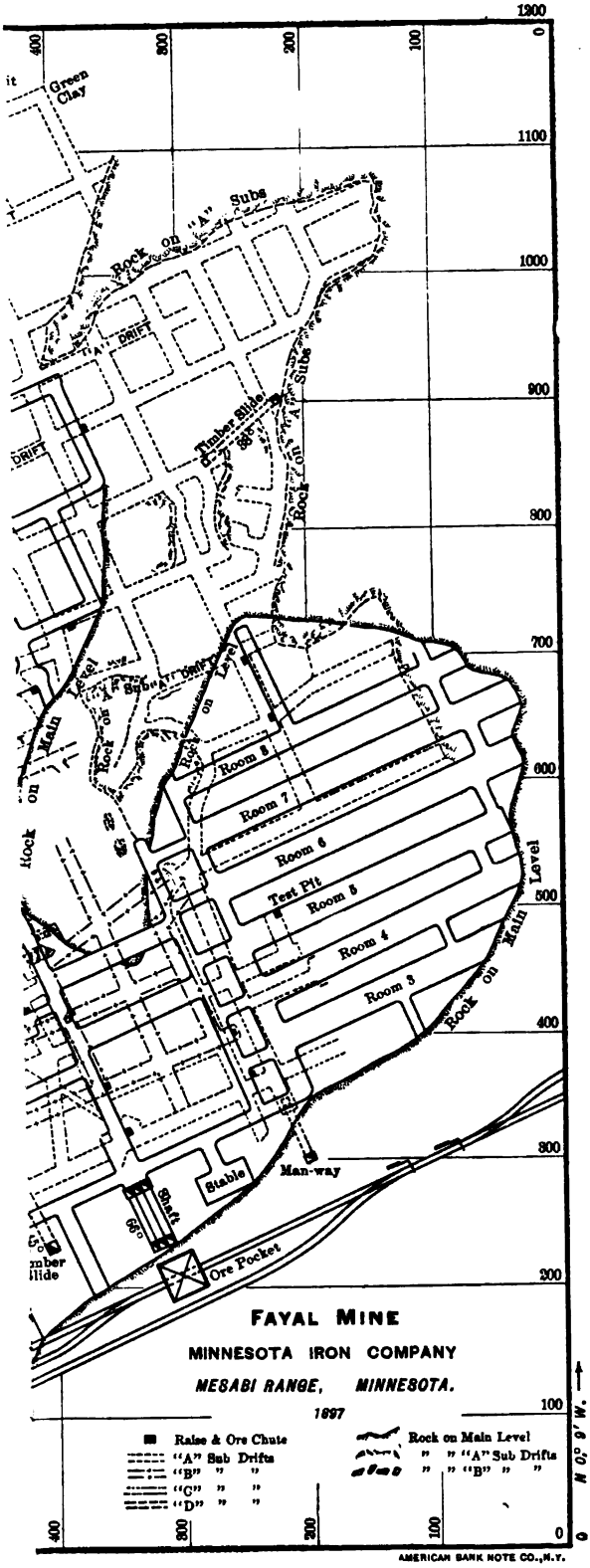
The Fayal Mine.—This mine, belonging to the Minnesota Iron Co., is the most extensively developed of the purely underground mines. Fig. 17 is a plan of the workings. I doubt if a more carefully and accurately laid-out iron-mine of equal size is to be found anywhere. The large area of developed ore is shown on the map; and there is another area, at least equally large, on the forty acres adjoining to the south, which is about to be developed.

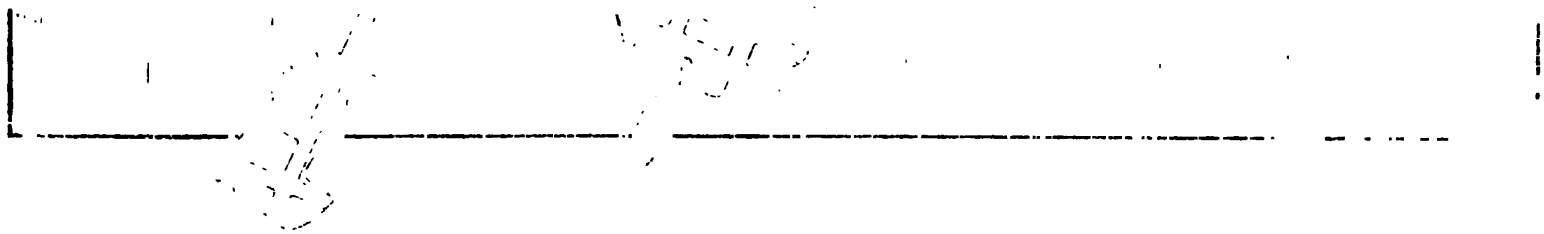
The Fayal shaft was sunk in the winter of 1894–5; in 1895 there were mined 125,000 tons of ore; and in 1896, 325,000 tons. All the ore is made into one grade, containing 62.5 per cent. of iron and 0.038 per cent. of phosphorus.

There is from 65 to 90 feet of surface over the ore. The top of the ore slopes south, as shown by the map. The method of mining has been by rooms, and by top-drifting and caving. The rooms are made about 24 feet wide, with pillars between of about the same width, or a little less. The rooms are carried up to the overlying sand, which may make them as high as 65 feet; the length may reach 300 feet. All rooms are filled, after they are completed, by running in the glacial drift that forms the roof, from the top of a raise put through to the surface for the purpose.

When rooming is not employed, raises are put up to the sand, and from the tops of these raises, what are called "sub-drifts" are put through the ore, dividing it up into a series of blocks or pillars, one drift-set high (see map). These "subs" are timbered with drift-sets. The first series of sub-drifts, just under the sand, are known as "A-subs." When the "A"-subs run into rock or the overlying sand, a second series of subs known as "B-subs" is run 12 feet lower, center to center, than the "A" subs. The "C"- and "D"- subs follow in order.

One series of subs is not extended under the one above, as a rule, until the upper one has been worked out; and, when necessary in order to reach timber or ore-chutes, the drifts are staggered so that the roof of the lower drift does not come entirely under the floor of the upper drift.


1200
1100
1000
900
800
700
600
500
400
300
200
100
0
Green Clay
Rock on "A" Subs
DRIFT
Timber Slide
Rock on "A" Subs
Rock on Main Level
Room 8
Room 7
Room 6
Test Pit
Room 5
Room 4
Room 3
Rock on Main Level
Stable
Man-way
Shaft
Ore Pocket
FAYAL MINE
MINNESOTA IRON COMPANY
MESABI RANGE, MINNESOTA.
1897
Raise & Ore Chute
"A" Sub Drifts
"B" " "
"C" " "
"D" " "
Rock on Main Level
" " "A" Sub Drifts
" " "B" " "
N. 9° 0' W.
AMERICAN BANK NOTE CO.,N.Y.



After blocking out the ore by these drifts, the pillars or blocks are mined by slicing and caving; *i.e.*, a drift will be run along the further side of a pillar; then this drift will be caved, and a second slice taken off alongside the first; and the operation will be repeated until the block has been mined. Each slicing-drift is not run entirely across the block; but two drifts are started, one from each side, to meet half-way down the side of the block.

Before caving these slicing-drifts, 1-inch boards of the cheapest grade obtainable are placed over the bottom of the drift, to keep the sand from mixing with the ore mined from the next lower series of drifts. The raises used for ore-chutes are placed about 100 feet apart; and, thus far, regular mine-cars, running on tracks, have been used in the sub-drifts. Fig. 18 shows the standard underground chute, car, etc.

The first main sub-drift timber-sets are made of 12-ft. caps and 8-ft. legs, making 10 by 8 ft. in the clear; and the sets used in slicing have 8-ft. caps and 8-ft. 8-in. legs, making 7 ft. 6 in. by 8 ft. in the clear. The top of the ore is irregular, a condition which often leaves bunches of ore above the top subs. This ore is obtained during the slicing and caving by raising up above the drift-sets and, if necessary, putting in "baby"-sets. When the next lower drifts are run under the caved ground, all of the 12 feet of ore is removed, as follows: The legs being 8 ft. long, the caps 1 ft. in diameter and the lagging 6 to 8 in. thick, a total excavation of about 10 feet is required to make room for the sets, and the space remaining between the top of the lagging and the planking above is filled with blocking, the planks being allowed to settle as little as possible.

At first the rooms were carried up by overhand stoping, with the aid of square-set timbering, in the usual manner; the square sets being 7 ft. 6 in. from center to center of legs, and the rooms therefore three sets wide. Recently, however, the experiment has been tried of putting in what are termed at the mine "stull-rooms." Fig. 19 shows the form of these rooms. The top of the room is first cut out by driving a wide drift just under the sand, and supporting this drift with saddle-back timbering, which becomes the roof of the room. This roof-timber is put in by driving from sub-drifts on the same level,

thereby avoiding the hoisting of timbers. The rooms could be started from the tops of raises if necessary. After the roof is thus securely supported, the ore is stoped under-hand,

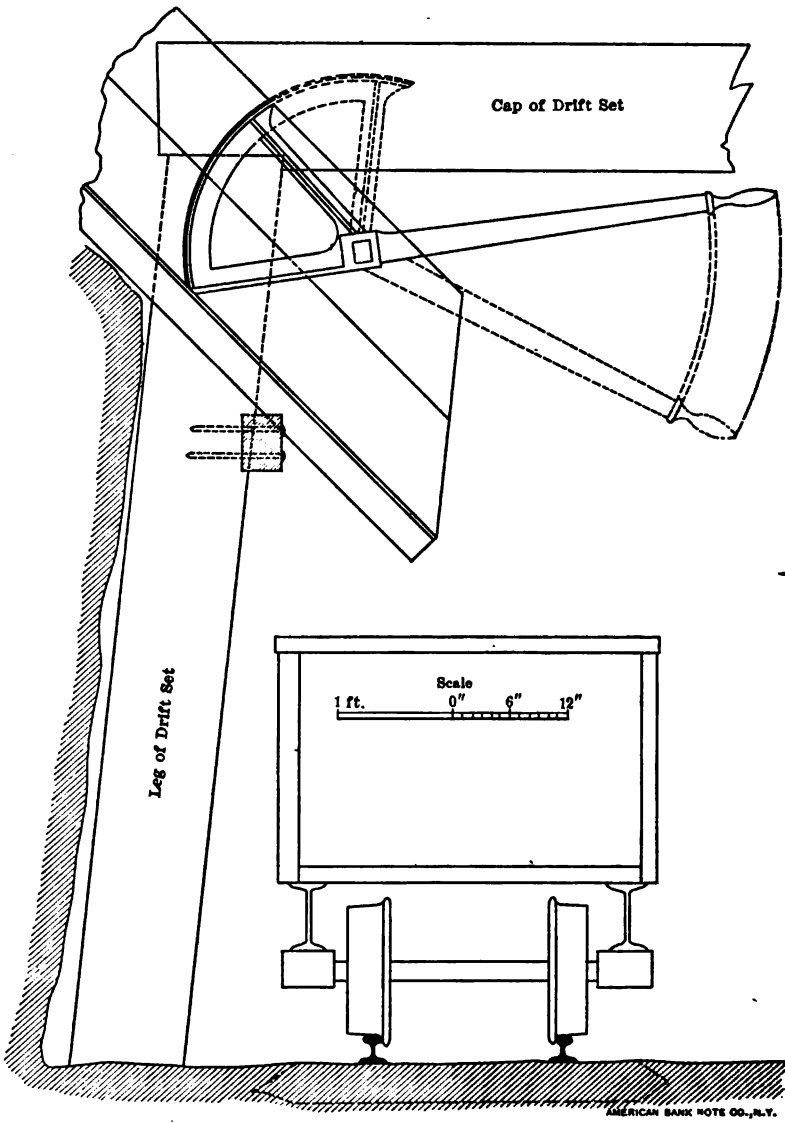


FIG. 18.—Standard Underground Chute used in the Fayal Mine.

through the raises, to the drift in the center of the bottom of the room, where it is run into the cars and trammed to the shaft. The sides of the room are left unsupported; and the

doubtful part of the experiment was, whether these sides would

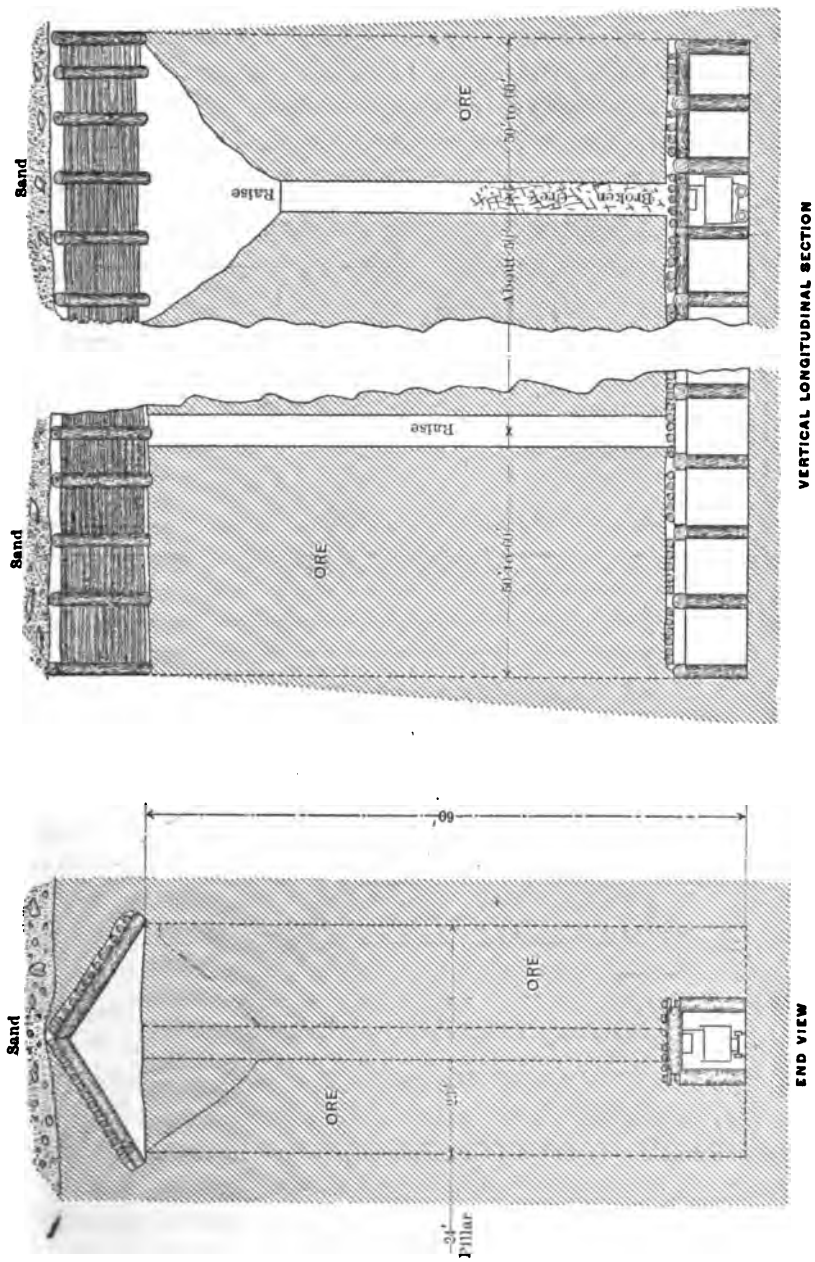


FIG. 19.—Stull-Rooms used in the Fayal Mine.

stand. A number of rooms have already been mined in this way without any trouble whatever; and, at least for the Fayal

deposit, the experiment seems to be successful. Many of the Mesabi deposits, however, are traversed by a system of parallel and almost vertical fissures or seams, filled with crushed quartz which, while only a fraction of an inch in thickness as a rule, would seriously interfere with this method. The Fayal has none of these seams, at least where the saddle-back rooms have

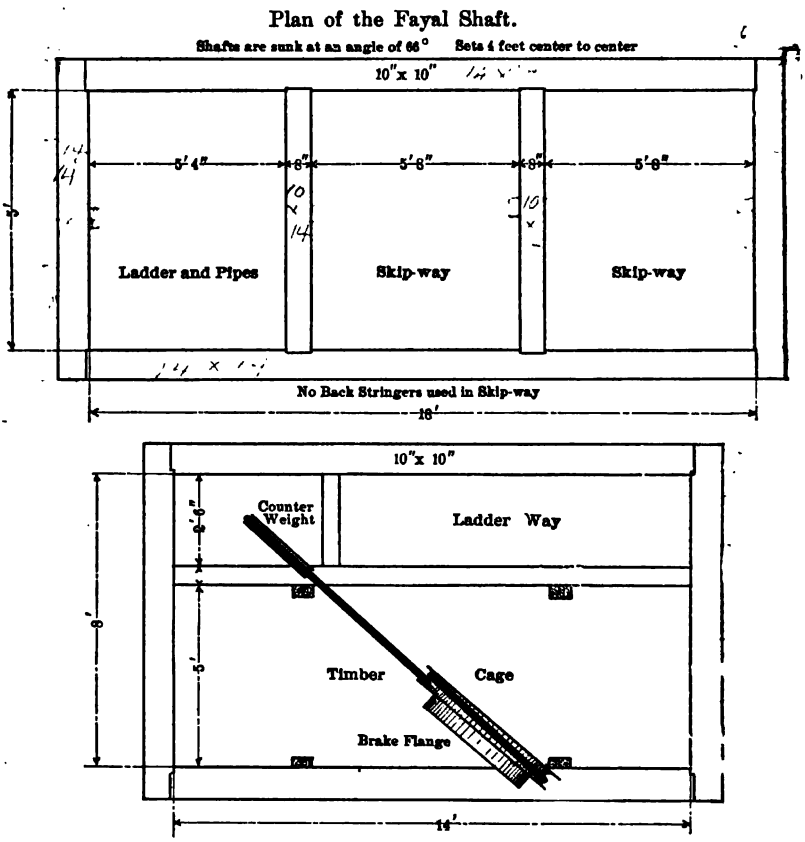


FIG. 20.—Plan of the Genoa Timber Shaft.

been made, and no trouble has arisen from caving sides. The ore obtained from these rooms is probably the cheapest ore obtained underground on the Mesabi, as the advantage of easy breaking is obtained with a low timber-cost.

The shaft is sunk in ore as close to the rock as possible, thereby locking up a minimum amount of ore in shaft-pillars. A plan of the shaft is shown in Fig. 20. It is sunk at an angle of 66° with the horizontal, at which angle the skips can be

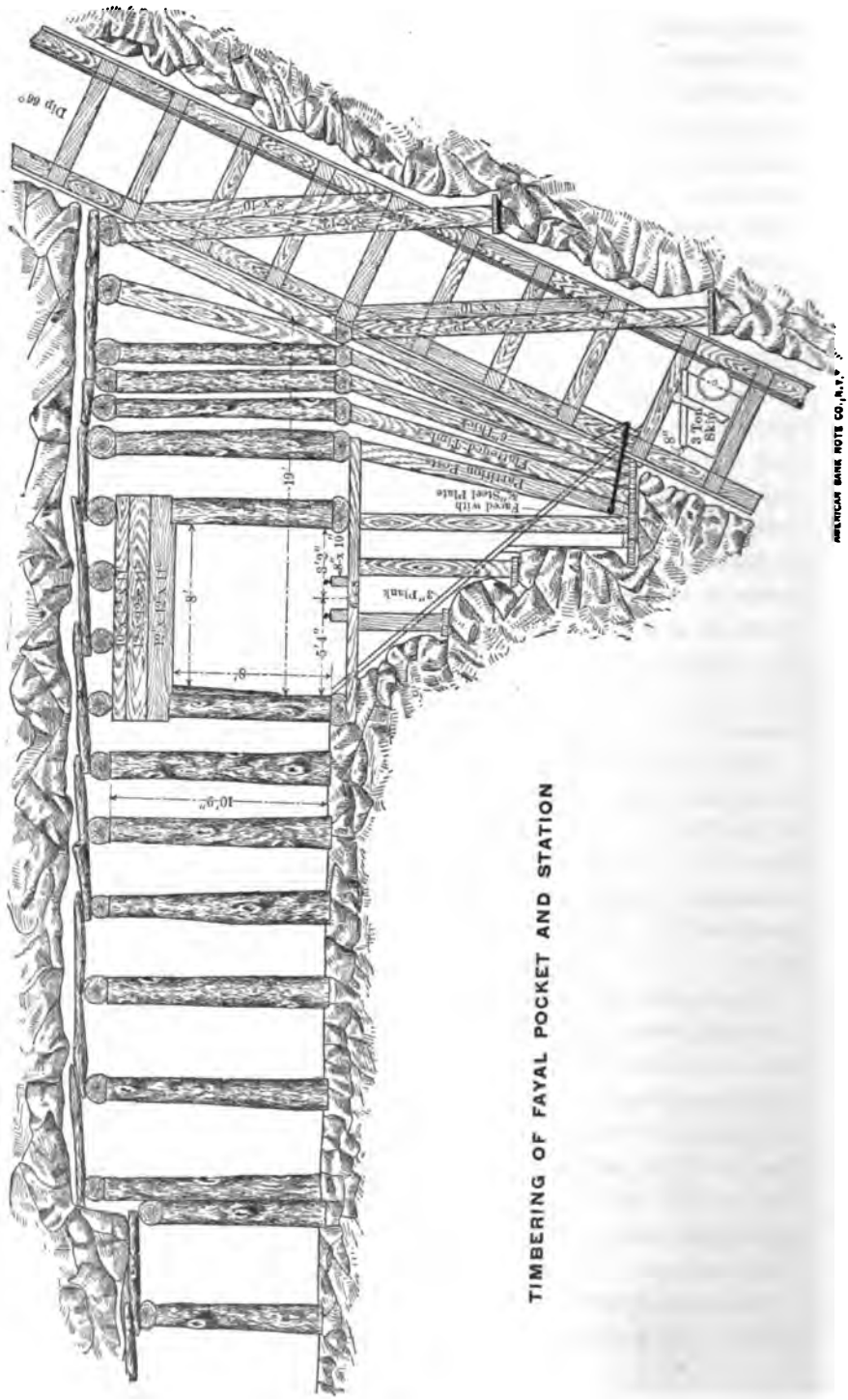
readily loaded from the underground pocket and dumped at the surface-pocket, without spilling the ore into the shaft. This angle does not require that back-stringers be used to keep the skips on the tracks. A section of the shaft and underground pocket together is shown in Fig. 21, and needs no special description. These pockets are being generally adopted, and make tramming and hoisting independent of each other for a short time.

Fig. 22 was taken looking towards the shaft, and shows the openings into the pocket. The shaft is cut off from view by a partition and doors used to diminish the draft. The tramming is done by hand in the sub-drifts, but for long tramming on the main level mules are used, and have been found to reduce the cost materially. The V-shaped hopper-cars used with mules are run over the pockets, and the doors are released and closed automatically. The hand-cars hold about 1½ tons and the mule-cars about 2 tons each. The skips hold 3½ tons. The wheels are made of manganese-steel, which is rapidly replacing all other kinds of material heretofore used for skip-wheels. The skips are operated in balance; and, as the shaft is used for hoisting only, its capacity is very large. An average output of 50,000 tons a month could probably be maintained at this shaft.

The Fayal shaft being an upcast, the head-frame is boarded in, to keep water from freezing on the rails and skip. The top of the Fayal pocket is 40 ft. and the head-sheave axis 65 ft. above the ground. The main timbers are 12 by 12 in. The maximum height of stock-pile is therefore 40 ft. At other shafts of the same company, the height of pockets is about 34 ft.

Stock-piles are made long and narrow, and are built up by dumping over the end, and not over the sides, as by so doing the stock-pile increases too fast to freeze hard. Stock-piles that have frozen during the winter cause a good deal of trouble in loading by steam-shovel in the summer. The hoisting-plant has two 18 by 42-in. cylinders, operated by Corliss valves without cut-offs, which are directly connected to a 6-ft. drum, fast upon the shaft. The plant was made by E. P. Allis. The hoisting-rope is a 1-inch Lang-lay rope, made by Roebling.

A special feature of the Fayal mine is the use of "timber-slides" for getting timber into the mine. Fig. 23 shows the



TIMBERING OF FAYAL POCKET AND STATION

FIG. 21.

bottom of one of these slides, which are made by putting up raises to the surface, 5 by 5 ft. in section, and equipping them with skid-ways, down which the timber is allowed to slide. The skid-ways are curved at the bottom, to diminish the velocity of the timber. The angle of the first slide was 45°; but the later ones were put at 38°, which is considered to be about the best angle.

The men are made to travel in and out of the mine through inclined raises specially prepared for them, so as to lessen the chances of fire in the main shaft.

The Genoa Mine.—This mine is just being developed. During the past year 17,136 tons of ore have been taken from the opening drifts. Fig. 24 shows the present condition of the mine. The execution of the work has been of the same high order as at the Fayal. The shaft is like those of the Fayal and Auburn. It is sunk on land owned in fee; but the first mining will be confined to ore held under lease. The bulk of the ore will come from the direction in which the shaft and main drift are pointing. No rooming has yet been done; but the ore is obtained from top sub-drifts, slicing and caving, as at the Fayal. The main sub-drifts have 12-ft. caps and 8-ft. legs, and the slicing-drifts will probably have the same height, but less width (8 ft.). The raises are placed 50 ft. apart for slicing and caving, instead of 100 ft., as in the Fayal. The subs are 12 ft. apart vertically, from center to center. The blocks of ore are about 50 ft. square when caving begins. The bottom is lagged with inch-boards before caving; cars are used in the subs; and the general work is similar to that done in the Fayal mine. The timber, however, is handled differently. Fig. 20 shows the plan of a vertical shaft especially designed for handling timber. The timber is cut on the surface at the timber-pile, loaded on trucks and wheeled to the shaft. The timber remains on these trucks until the place is reached in the mine where it is to be used. Gravity is utilized to operate the timber-cage. By altering the weight of the balance, injured men, cars, etc., may be brought up on this cage more conveniently than in the main hoisting-shaft.

A detail of some interest, common to the Fayal, Auburn and Genoa mines, is the use of special rooms underground for thawing dynamite. These rooms are generally located at the

end of a special drift, as shown on the map of the Genoa, Fig. 24, and are kept warm by steam-heat. Each room contains lockers, one of which is assigned to each contract-party. No candles are allowed in this room, on account of the danger of fire; and, for the same reason, the men are required to travel through the ladder-way of the timber-shaft. The hoisting-plant is a duplicate of that at the Fayal.

Hibbing.—At Hibbing is a group of mines operated by the Lake Superior Consolidated Mines, of which the Burt, Rust and Hull mines are specially interesting. They are being very thoroughly developed, on lines differing somewhat from those thus far described; but unfortunately it was impossible to secure maps of them in time for publication. The main drifts are opened on the diagonal plan; and it is proposed to use a system of rooms with square sets. The rooms are to be caved, and the pillars mined by some other method. The novel feature of the proposed method is the arrangement of the rooms. These are put in first at the boundary of the working-area of the shaft; and as these rooms are completed and caved, new ones are opened nearer the shaft.

At Hibbing is also the well-known Mahoning property. The Mahoning Company owns a very large area of ore, and will become in the future one of the heaviest producers on the range. Thus far mining has been confined to steam-shovel work in one cut; but the rest of the property has been very thoroughly and systematically explored. Fig. 25 shows one of the Mahoning shovels of the latest design.

Discussion of Different Mining Methods.

It will be seen from the descriptions and maps of the various mines that there are several important points in which the methods of mining differ. Perhaps the first to attract attention, had the maps of the mines at Hibbing been published, would have been the method of laying out the main haulage-drifts.

Haulage-Drifts.—The Lake Superior Consolidated Mines follows the method of diagonal drifts, while the Minnesota Iron Co. has adopted the rectangular plan shown in the map of the Fayal mine, Fig. 17. The latter company used the diagonal plan for the first level of the Auburn mine, but abandoned it

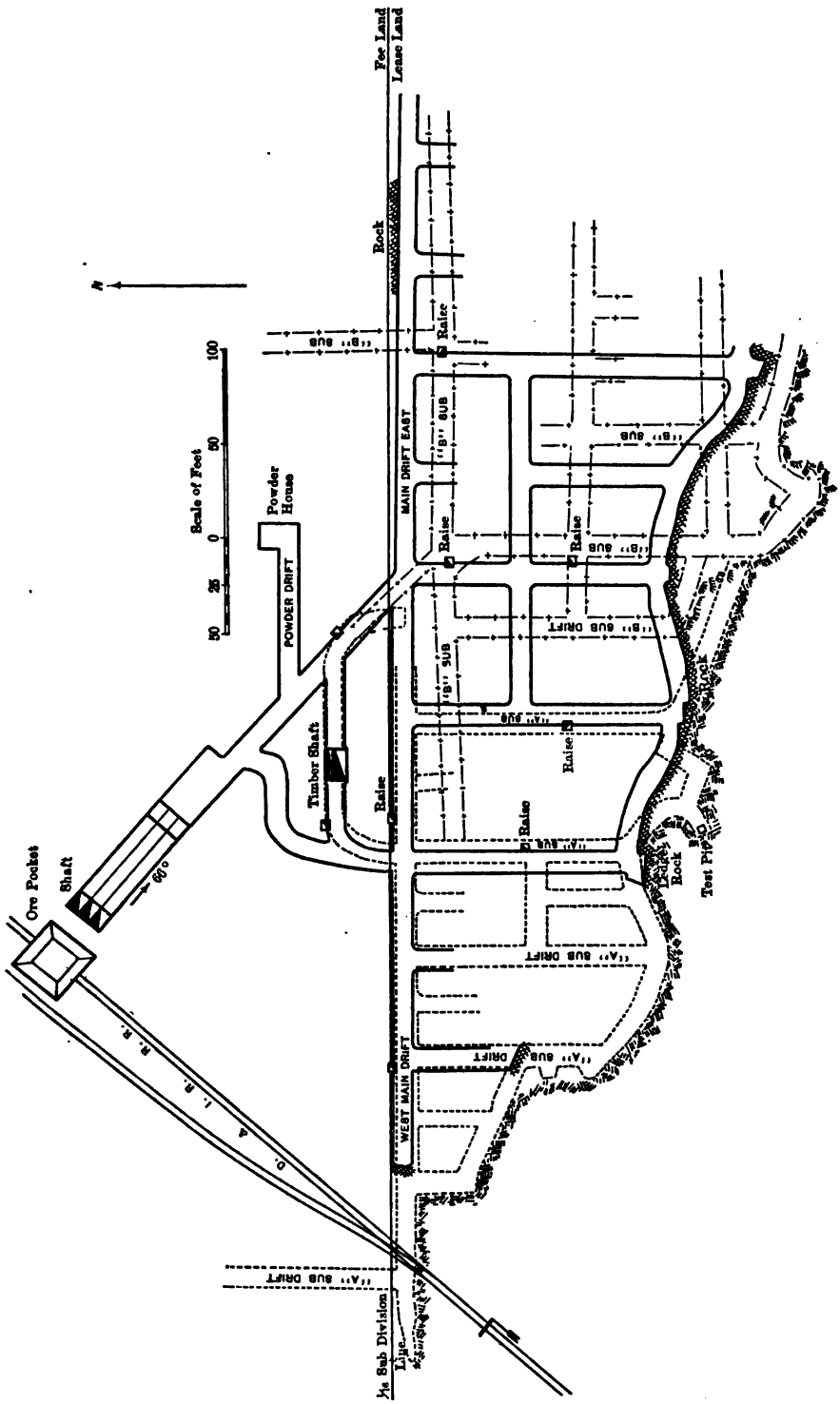


FIG. 24.—Genoa Mine, Minnesota Iron Co., Mesabi Range.

in driving the second level. The advantage of the diagonal plan lies in the facility it offers to tramming, since the distances from the working-places to the shaft are shorter and more direct, and all curves are very easy. Its disadvantage is that the blocks of ore formed by the drifts are diamond-shaped, and if the sub-drifts, from which caving and slicing are done, are placed directly over the main drifts, it is more difficult to mine all of the ore than if the blocks were rectangular. It would be very difficult to slice away and cave a system of sub-drifts laid out on the diagonal plan, on account of the triangular pillars that would be left at certain stages of the work; and furthermore, it would be much more difficult for the men to keep track of the ore, and to know just when all of a pillar had been mined, since the angles between drifts would not be right angles. The rectangular plan for sub-drifts, if rigidly maintained, causes less loss of ore from these sources than the diagonal.

Still, the saving in tramming with the diagonal arrangement must be marked; for it has become a settled principle that, in cases of large traffic over large areas, diagonal roads must be obtained at any cost. Many of the Mesabi ore-bodies cover large areas, and, as in the Fayal, the area tributary to a single shaft may reach 40 acres. In such cases it would seem to be advantageous to have a few roads radiating from the shaft. In the case of the Auburn, the area covered is not large enough to make any difference.

It might be practicable, although no case is known to the writer in which it has been tried, to combine the diagonal and rectangular systems in those mines using top-slicing and caving methods. A little trouble might be experienced in getting out the ore of the main level; but it does not seem that this could offset the advantages obtained in the tramming.

The advantages of the diagonal system will probably be more marked when mining has nearly reached the main drifts, and repairing has to be done to them.

Rooms.—A second point of difference is in handling the square-set rooms, after they are completed. At some of the mines the rooms are caved by blasting in the roof and the timbering, while in others they are filled by running in the overlying drift through raises put up for the purpose.

There can be little doubt, after all the experience that has been had with rooms on Lake Superior, that rooms cannot be caved without incurring large losses in the pillars, unless the latter are two to three times as wide as the rooms. When pillars and rooms are of the same width nominally, the pillars are actually about 2 feet narrower than the rooms; and when the rooms are caved, the top of the pillar is broken and lost, and the rest of it is so crushed as to make regular mining in it impossible. All that can be done is to "run" the pillar until the sand comes.

Square-set rooms, which have been so largely abandoned elsewhere on Lake Superior, have been revived on the Mesabi range, and one naturally asks why this is the case. Numerous causes have probably operated to bring rooms into favor again. The main one is, of course, that ore mined from rooms is the cheapest ore obtained underground. Gravity assists both in the breaking of the ore and in the loading into cars; and the only increased cost is in the timbering. As has been shown, the timber-cost has been reduced to a small amount at the Fayal mine by the use of saddleback timbering; and in those mines which are compelled to use the old form of square sets, the timber-cost is less than on the old ranges, by reason of the cheapness of timber on the Mesabi range and the use of larger sets.

In spite of these advantages, however, it is safe to predict, in the writer's opinion, that history will repeat itself, and that rooming will soon be abandoned as a regular system of mining in this region. Its disadvantages are not developed until the wrecking of the surface has become general, and the removal of the pillars begins. It is then that the increased cost of ore from pillars, the uncertainty of the work, and the accompanying large loss of ore, force the conclusion that while rooming is highly satisfactory as regards the cost of ore directly taken in that way, it is highly unsatisfactory as regards the cost of getting the ore that is left, and the average cost for the whole deposit. When the capping of the ore is solid rock, the above remarks do not apply; but on the Mesabi the covering is sand and glacial drift.

A third interesting difference in the methods of mining, which has not been brought out in the descriptions given, is the

manner in which bunches of ore, lying above the first or top drifts in the top-caving system, are mined. In the description of the Fayal mine, it was observed that such ore would be taken out on the retreat, when caving began, by the use of small or "baby-sets" above the main sets. This system, however, has not yet been fully developed, and its ultimate success is uncertain. At the Franklin mine, near Virginia, the regular top-caving system is employed, the raises being about 30 ft. apart and 3 ft. in diameter. Wheelbarrows are used. The main timber-sets have 8-ft. legs and 5-ft. caps, so that about 10 ft. of ore are taken in one slice. On the top-slice, however, the legs are cut to suit the height of ore at each particular place, the caps having a constant length (5 ft.), while the legs may be of any length up to 14 and 16 ft. This method has been well developed, and is working very successfully. The use of the wheelbarrows and the placing of the raises so close together does not necessitate the maintenance of level drifts, but the drifts are sloped somewhat to suit the dip of the top of the ore. In strong contrast with this is the arrangement at the Fayal mine, where raises are 100 feet apart and cars are used on the top-drifts, and the drifts are large and driven with as much care as the main-drift leading to the shaft. It will be interesting to see which system prevails ultimately.

The extent to which lagging is used varies much. In some of the mines the sides of rooms are lagged even when the rooms are to be blasted in. In others, no side-lagging at all is used in rooms, even when they are filled. This is perhaps a small difference; but it is interesting.

In top-slicing and caving, all cover the bottom of the drift, before caving, with boards, split lagging or slabs, while some lag the side next the ore as well, and others do not. On the last point it makes a difference whether it is the top or first slice, or a lower one. The first slice is taken directly under the unbroken sand, and this sand does not follow the ore very closely. When a pocket of sand is tapped, however, or when caving is proceeding under ground already broken up, the sand and drift rush up to the side of the ore; and under such conditions side-lagging would seem to pay.

None of the mines follow the system employed at the Chandler, of leaving ore in the back of the drift, to be caved with the

covering on the retreat. The reason is that the grade of Mesabi ores is not high enough to stand the loss in percentage of iron which always accompanies this system, more or less. Ore obtained by such a method is, of course, much more cheaply broken than that which is sliced or drifted out, but it is a question of both gain and loss.

Conclusion.

Mining on the Vermilion range has reached a settled, fixed system, and ranks with the best practice on Lake Superior. It is not probable that any radical change will be made in the future, as the present methods are the results of considerable experience and the conditions arbitrarily limit the methods of mining.

On the Mesabi range, however, one is forced to the conclusion that no company is yet satisfied with its method of underground mining. Experiments are being tried constantly; schemes are talked of continually; and, with the best talent on Lake Superior at work on the problem, it is possible that in the near future some new system will be developed which will be considered the best. Thus far the practice has been to apply to the new conditions the well-known systems of the older ranges, from which those directing the work came. It is difficult, however, to see whence any radical change of present methods can come, except in the development of open-pit mining.

In a previous paper* the writer attempted to determine the difference in the cost of mining ore underground and in open-pit, exclusive of stripping-cost, and placed it at 21.5 cents per ton.

Although the data from which this difference was computed were not as extensive as one would wish, it is still believed to be a conservative estimate, and one may reasonably expect open-pit work to become much more highly developed in the future than it is at present.

The cost of opening and certain risks would be increased, perhaps, by open-pit mining; but the advantages obtained would seem to justify such additional expense and risk.

* "Open-Pit Mining with Special Reference to the Mesabi." *Proceedings of the Lake Superior Mining Institute*, vol. iii., 1896.

Thorough and accurate preliminary exploration, the importance of which is becoming more and more recognized, would remove all risks.

The writer acknowledges his indebtedness to nearly all of the mine-managers and superintendents of Minnesota, who have afforded him every courtesy and much assistance in collecting the data given in this paper. He is specially indebted to Mr. D. H. Bacon, President of the Minnesota Iron Co.

The Electrolytic Assay as Applied to Refined Copper.

BY GEORGE L. HEATH, SOUTH LAKE LINDEN, MICH.

(Lake Superior Meeting, July, 1897.)

It may at first appear doubtful that any further ideas can now come from such a well-trodden soil, when we consider that the ground of the subject has been so thoroughly gone over in many of its phases and that observations have been recorded in technical journals, ever since the days when Luckow and his cotemporaries brought the electrolytic deposition of pure copper to the position of an exact scientific method. Certain facts, however, which seem worthy of note, have come under the writer's observation during an extended use of the method in a large refinery, involving frequent tests of refined Lake and electrolytic copper, and thousands of assays of native copper concentrates and tailings.

This work has required a careful study of the conditions suited to the assay of refined copper, and of ores and tailings—conditions which must be varied for each particular case to secure the best results. Especially is this true when we come to the exact determination of pure copper in the high-grade refined metal of the present day.

Very slight traces of impurities (especially tellurium, selenium, bismuth, tin, arsenic and antimony) seriously injure the properties of copper, notably its electrical conductivity.

The increasing demands of electrical science and construction lead to the conclusion that the analysis of all high-grade American coppers, at least, has become, and must be in future (in or-

der to be of any value to an intelligent manager), a matter of accuracy, permitting no greater limits of variation or error in the determination of copper than 0.01 per cent., and in the estimation of foreign impurities, 0.001 per cent., and even less.

This is evident when attention is called to the fact, recently noted by A. S. Warren,* that American high-grade refined copper contains about 99.9 per cent. of pure copper, including the variable trace of silver present, which is best deposited with the copper, the amount of silver having been ascertained by a special assay. This allows only 0.1 per cent. for oxygen, which constitutes the greater part of the remainder, and for traces of other impurities, which vary in nature and amount with the perfection of the refining and the source of the metal, whether from native mineral or sulphide-ores.

Oxygen is not now reckoned as a harmful impurity, since within a low limit, easily maintained, a small amount is apparently necessary to insure a proper "set" and good physical structure. It is, of course, an absolute impossibility to pour copper in direct contact with air, and not have a little oxygen present in the cast metal.

The electrolytic assay is found to be the only chemical method sufficiently accurate for the estimation of the percentage of copper in any high-grade manufactured product, though there are other methods which, as noted by western assayers, are better suited to complex ores.

Most text-books and miscellaneous articles on the assay of copper deal with its application to ores and with separations in commercial alloys, and are deficient in directions for the very exact assay of 99.9 per cent. material.

With the exception of Fresenius and Hampe's methods for the complete analysis of copper, the directions usually given by writers on assaying call for a little sample of 0.1 gramme to 0.5 gramme in weight.

To permit the attainment of check-results agreeing within 0.01 per cent., at least 5 grammes of copper must be taken for each assay: (1) because the sample must be an exact average of a bar or plate, which may not be strictly homogeneous; and (2) because the best laboratory-scales do not weigh closer than 0.0001 gramme with certainty.

* *Eng. and Mining Journal*, April 10, 1897, p. 351.

If only a 0.5 gramme sample is taken, a difference of 0.0001 gramme, from any cause, would change the result 0.0001 ÷ 0.5 = 0.02 per cent.—too great an error.

I believe 5 grammes of copper, except in special cases, is the best weight to assume as a sample of high-grade metal, since it may be deposited from a solution of small bulk (which is desirable) and yet is a sufficient quantity to give the required accuracy in weighing.

Before detailing the method used by the writer for this special class of material, a few suggestions may be made as to the apparatus, etc., which have proved most satisfactory for a large amount of electrolytic work.

Electrodes.—I favor the use, for all copper analysis, of the platinum spirals and cylinders (preferably slit open on one side) since they are less expensive than dishes, more easily removed from the solution, the plates of deposited copper are cleaner, and the copper may be deposited from a solution of an ore or mineral, without filtering out the siliceous gangue.

The cylinders may be 2 inches (5.1 cm.) high, and 3½ inches (8.9 cm.) to 4 inches in circumference, and the wire stem by which the electrode is clamped to the holder should extend at least 2 inches (5.1 cm.) above the cylindrical plate. The total weight is 11 to 14 grammes.

Stands.—These, with the attached clamps, should be easily adjustable to beakers and dishes of varying height.

A cheap apparatus designed by the writer fulfills these conditions and is easily constructed.* Other good forms are described in the works of Classen,† Smith‡ and Dr. Peters.§

The Source of Electricity.—The crawfoot or "gravity" cell is the cheapest reliable primary battery for a moderate amount of general work, not warranting an expensive plant. At least, that much-maligned current-generator may be made to give satisfaction if it is very frequently overhauled—at least twice a month when in constant use, or as often as it becomes choked with deposited material at the bottom. It does not pay to use the zincs up too much; but about half the expense may be saved

* *Jour. Amer. Chem. Soc.*, June, 1896, vol. xviii., p. 558.

† *Quantitative Chemical Analysis by Electrolysis.* Am. ed., pp. 48-54.

‡ *Electro Chemical Analysis.*

§ *Modern Copper-Smelting*, 7th ed., p. 45.

if a mould is provided, and the zincs are cast with the spent fire at the close of a day's work at the wind-furnace. The old zincs may be scraped, broken up, and re-cast with twice the weight of new spelter. Melt the metal at low heat, and use a covering of charcoal in the graphite crucible.

A device, the good effects of which have possibly not occurred to all, consists in refilling the cleaned battery-jars one-fourth to one-third full of old battery-solution (saturated zinc sulphate) and then diluting with water till the zincs are covered.

Pure water, alone, offers a great resistance to an electrical current; and much zinc must be dissolved unnecessarily to establish the proper conditions for maximum current, unless old battery-solution is introduced as just recommended, when the cell will be found almost immediately to be in good working order.

By far the best current-generator is the dynamo with incandescent lamps in circuit, preferably combined with a small plant of storage-cells at those laboratories where the current from a dynamo is available only at night. The dynamo-current may at that time be caused to run the assays and charge the storage-cells for use during the day.

One of the best storage-cells is the "chloride accumulator" (type 5 E, Electric Storage Battery Company, Philadelphia) costing about $10 each. A voltmeter is necessary; and a Weston instrument (0 to 5 volts) may be obtained for about $25.

These cells may be arranged in independent groups of three, in series such that one group, or series, may be discharging while another set is being charged. Each cell, when charged, has a potential of about 2.5 volts, and three cells in series will give from 9 to 10 amperes for from 14 to 15 hours. The strength of the current may be easily varied by passing it through a rheostat.

A cheap and efficient form consists of 500 feet (152 m.) of iron wire stretched on a wooden frame about 6 feet (1.8 m.) long, and fastened to the wall behind the assays.*

Arrangement of Analyses.—These should be connected in parallel with the current so that each assay may be independently connected, or withdrawn, without stopping the current passing through other copper-solutions.

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* Smith's *Electro Chemical Anal.*, 1st ed., pp. 28–29 (2d ed., pp. 33–34).

If a pair of main wires are led along a shelf or frame, the stands carrying the clamps for the electrodes may be attached by small short tap-wires with soldered joints.

Acidity and Concentration.—There has been much needless disagreement concerning the kind of acids and the proportions of the same which should be present in a solution of copper undergoing electrolysis.

Chemists have recommended both dilute nitric and dilute sulphuric acid as the menstruum and solvent; but generally no sufficient reason has been given why one acid should be preferred in the separation of copper.

One property of nitric acid, or a nitrate, differentiates its behavior during electrolysis from that of sulphuric acid, namely, its conversion to ammonia by the electric current.

This change* may be complete if sufficient time be given, and some of the acid chemically combined with iron, for instance, may be attacked, the whole reaction possibly resulting in the deposition of foreign oxides upon the copper plate. This reaction is the cause of nearly all the trouble in copper-assaying with nitric acid solutions.

I have proved repeatedly with rich and lean cupriferous material that fouling of the electro-deposited copper need not occur (even in the presence of considerable amounts of iron, alumina, titanium, cobalt, nickel, zinc, lead or manganese, etc.), if the strength of current is only just sufficient to deposit the copper in a reguline plate (0.05 to 0.1 ampere for ores, and 0.2 to 0.3 ampere for refined copper), and if also sufficient sulphuric acid is present:—

(*a*) To keep all foreign bases in solution as sulphates after the deposition of the copper on the cathode.

(*b*) To form hydrogen ammonium sulphate with all the nitric acid originally present, but afterwards decomposed by the current.

(*c*) To provide a small excess of free sulphuric acid in addition. This excess of free acid should be increased with increase of current, and according to the proportion of other bases dissolved with the copper in the solution undergoing electrolysis.

If arsenic or antimony occur in copper in appreciable quan-

* Classen's *Analysis by Electrolysis*, p. 3.

tity they should first be removed by evaporating the solution with chlorine, or bromine.* F. L. Sperry's recent paper† does not apply his knowledge of the behavior of nitric acid to the preparation of the analysis, since he directs the solution of 1 gramme of copper-ore in 25 cc. nitric acid, the evaporation of the liquid to dryness with only 10 to 15 drops of sulphuric acid and the final addition of a comparatively large amount (7 cc.) of nitric acid (sp. gr., 1.2), and 4 drops of sulphuric acid.

He writes that the electrolysis must not be too long continued, or nickel, etc., will be deposited. No wonder, when the proportion of nitric acid is so large that its complete change to ammonia means nothing less than the final neutrality, or even alkalinity of the solution.

The same chemist has shown that copper is deposited in a spongy condition from sulphuric acid in large excess. Titus Ulke‡ recommends the use of 1.25 cc. of strong sulphuric acid, and the final addition of only 6 to 8 drops of nitric acid. The writer has found that whether a little nitric acid is added or not, if the sulphuric acid is in sufficient excess to keep the liquid always strongly acidified, the separation of copper and nickel is always complete. Witness the following check-analyses on two small samples of nickel-ore from Sudbury, Ont.:

Check-Analyses.

Sample No.		I.	II.
Volume, cc.,		100	100
Current, amp.,		0.3	0.15
Weight, gramme,		1.	1.
Copper, per cent.,	*a.*	13.54	13.67
	b.	13.39	13.68

The plates in I. were slightly tarnished from excessive current, but all appeared nickel free. In II., the ore was decomposed with 15 cc. nitric acid, the nitric acid removed by evaporation with 5 cc. sulphuric acid (sp. gr., 1.84) and after dilution 0.3 cc. strong nitric acid was added, a minute amount, supposed to be nearly equivalent to the copper existing as sulphate. The deposited copper was not spongy.

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* Smith's *Electro Chemical Analysis*, 1st ed., p. 93 (2d ed., p. 112).

† In Peter's *Modern Copper-Smelting*, 7th ed., p. 44.

‡ *Eng. and Min. Jour.*, Jan. 30, 1897, p. 114.

Nitric acid works very well alone for separations only when nearly 5 per cent. of free acid is always kept in solution, or enough to make the liquid always strongly acid; but in dealing with concentrated solutions of refined copper it is best to use a minimum amount of a carefully proportioned mixture of the two acids mentioned.

Tests for the Completion of the Electrolysis.—Some authorities recommend that the solution be diluted with water when the assay is nearly finished, and say that if no copper makes its appearance on the clean part of the platinum electrode, the plate may be removed.

This is a misleading test with high-grade material, at least, since a trace of copper (0.001 gramme or less) frequently remains dissolved for twenty-four hours or more after the solution is perfectly colorless, and until the nitric acid, if present, is nearly all converted to ammonia. It is well to hasten the removal of this trace by an increase of current, if the copper analyzed is very pure.

Tests for Copper with Hydrogen Sulphide Water and with Potassium Ferrocyanide.

Indicator.	Copper Solution Taken.	Copper.	Total Volume, with Reagents.	Interfering Elements Added.	Color-Reaction.
	Cc.	Grammes.	Cc.		
H_2S water...	0.4	0.000005	2	None.	Good perceptible brown tint.
K_4FeCy_6.....	0.4	0.000005	2	None.	Good reddish brown tint.
H_2S water...	1.0	0.0000125	2	None.	Dark brown.
K_4FeCy_6.....	1.0	0.0000125	2	None.	Dark reddish brown.
H_2S water...	0.4	0.000005	2	Chloride of iron	Chloride masks color. Sulphate of iron has little effect.
K_4FeCy_6.....	0.4	0.000005	2	Chloride of iron.	Iron interferes; blue color.
H_2S water...	0.1	0.0000012	2	None.	Distinct though faint tint.
K_4FeCy_6.....	0.1	0.0000012	2	None.	Nothing perceptible.

Hydrogen sulphide water and solution of potassium ferrocyanide have been recommended as better means for detecting traces of copper; and the foregoing table records experiments

with these reagents upon standard copper-solution. The measured portions of solution of copper sulphate (1. cc.= 0.000012 gramme Cu) were placed in the cavities of a white porcelain tile, such as is used with volumetric titrations.

According to these tests, 1 cc. of a liquid which gives absolutely no tint of brown with H_2S must contain much less than 0.000,001 gramme of copper; and H_2S will detect copper in presence of iron, or a trace of arsenic or antimony.

Method for Analysis of Refined Copper.—It is presupposed that a complete analysis is not required, and that the trace of silver usually present is to be deposited also with the copper, since the silver can be more accurately estimated by a special test on a large sample.

Dissolve about 5.0 grammes of the metal in a tall, narrow No. 1 lipless beaker 1¾ inches (4.4 cm.) in diameter, and 4¼ inches (10.8 cm.) high, in 20 cc. of nitric acid (sp. gr. 1.42). Evaporate, after removing the cover, with 5 cc. of sulphuric acid (sp. gr. 1.84) until the mass is dry, or the white fumes are seen. (The above-mentioned quantity of sulphuric acid is calculated as the least possible excess over the 4.25 cc. necessary to form cupric sulphate.) Cool the beaker; add 2.7 cc. of nitric acid, which can never neutralize, if reduced to ammonia, more than the 4.2 cc. of sulphuric acid; dilute with 30 cc. of water; and heat till all the copper sulphate is dissolved. Dilute to 120 cc.; place the beaker in position on the electrolytic stand; introduce the platinum electrodes, which have been previously cleaned with acid, washed with water and absolute alcohol, ignited and weighed; and finally turn on the electric current, covering the beaker with glass to prevent loss by spattering.

The body of the cylindrical electrode should now be just covered with the liquid and any little liquid lost by evaporation should be replaced from time to time.

Electrolyze this concentrated solution with a current of 0.25 to 0.3 ampere (from storage-cell or incandescent lamp), that is, as rapidly as possible without producing sponginess in the deposited metal.

The current of one old-style, 16 c. p. lamp, 75 volt circuit, or a series of 6 "gravity" batteries, suffices for two of these assays connected in parallel.

When the solution has appeared perfectly colorless for at

least 12 hours, remove one cc. to the cavity of a white tile, and test with one cc. of freshly prepared hydrogen sulphide water. Continue the electrolysis until the test-portion remains absolutely free from any brown tint when compared with clear water.

The sets of weights used for this work should be very accurately and carefully compared with each other, since slight discrepancies would prevent extreme accuracy in weighing.

When the deposition is strictly complete, hold the stem of the electrode tightly with one hand, while the clamp is unscrewed with the other, then instantly raise the electrode and plunge it into a large beaker of cold distilled water. If it is preferred to have the assay resting on a block of wood, remove the block with one hand and instantly drop the beaker and replace it with the beaker of cold water before the screw is loosened. Complete the washing with water; then wash with absolute alcohol; ignite the film of alcohol adhering to the plate; cool a short time and weigh.

It is not necessary to dry in a desiccator, if the electrode is allowed to remain in the balance-case a few minutes before taking the weight. As the total weight of the cathode (platinum and copper) is only about 20 grammes, any error due to the difference in volume of the platinum and the brass weights is negligible.

That the electro-deposition of copper may be made strictly complete and suited to the assay of the highest-grade metal is easily proved by testing the colorless liquid with sulphuretted hydrogen water, and has been further demonstrated:

1. By the assay of miscellaneous specimens of American metal, in which all the foreign metallic and non-metallic elements have been estimated by separate analyses. The sum total should then come within one or two hundredths per cent. of one hundred, if the sampling is accurate.

2. By analyzing some very pure cathode-metal already once deposited as a sheet from a solution of a pure Lake copper. This sheet was cut into a strip and drawn cold through a die into wire, to remove the gases by mechanical work. It was then annealed at low heat, and the surface of the wire was carefully cleaned and polished. This copper was found to be free from traces of any other elements, except a possible trace of silver

Tests of Electrolytic Analysis Upon Specially Prepared Pure Electrolytic Cathode-Copper.

No.	Weight Taken.	Total Volume.	Time of Electrolysis.	Oxygen: *d* = Determined, *c* = Calculated by Difference.	Other Elements.	Copper (Including Silver).	Averages.	Totals.
	Grammes.	Cc.	Hours.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
1	1.0	100	36 to 48	(*c*) 0.0521	0.026	99.934 99.902	99.920	100.00
2	1.0	100	36 to 48	(*c*) 0.0570	0.023	99.921 99.923	99.92	100.00
3*a*	1.0	100	30 to 40	(*d*) 0.048	0.015	99.89? 99.904 ? 99.920 99.83? 99.936	99.92	99.983
3*b*	5.0	120 to 135	42 to 54	(*d*) 0.048	0.015	99.905 99.914	99.91	99.973
4	1.0	100	36 to 48	(*d*) 0.049	0.015	99.913 99.921 99.924	99.92	99.984
5*a*	1.0	100	36 to 48	(*d*) 0.099	0.083	99.790 99.80 99.794 99.795	99.80	99.982
5*b*	5.0	120 to 140	40 to 54	(*d*) 0.099	0.083	99.815 99.801	99.81	99.992
6	5.0	120 to 140	40 to 54	(*c*) 0.054	0.006	99.943 99.939	99.94	100.00
7	5.0	120 to 140	50 to 60	0.000	0.000	100.002 99.992	99.997	99.997

(see No. 7). A few results are tabulated, and show a gain in accuracy of sampling and agreement of check-results when 5-gramme samples are assayed instead of 1-gramme or less.

In samples 3, 4 and 5, the oxygen was very closely determined by Hampe's method; but assays 3*b*, 4*b* and 5*b* could not be cut from the very same portions from which 3*a*, 4*a* and 5*a* were taken.

Considering the possible slight difference of sampling involved, the tests on the three samples mentioned agree well enough; and No. 7 is exact.

Tests on 5-gramme samples check a little better, and are far less liable to be slightly low from the retention of a faint trace of copper (perhaps 0.0001 gramme) in solution, which loss would, as before stated, have no perceptible effect on the percentage determined upon a large sample.

The solvent power of nitric acid upon a deposited plate of copper in a concentrated solution is very marked; hence that acid could not well be used alone for refined copper; but by the expedient of removing the nitric acid by sulphuric, and then adding a small measured amount of the nitric, we produce a solution which shall contain at the end of the electrolytic operation only ammonium bi-sulphate and a little free sulphuric acid, which keeps impurities dissolved but does not have appreciable solvent effect on the copper while suspended in the liquid, or during its subsequent removal and washing.

The method detailed will, if carefully followed, ensure complete deposition of copper in a pure condition from solutions of refined metal, or of any other metallic product, when harmful elements are not present to interfere with electrolysis.

A Mine-Dam.

BY WILLIAM KELLY, VULCAN, MICH.

(Lake Superior Meeting, July, 1897.)

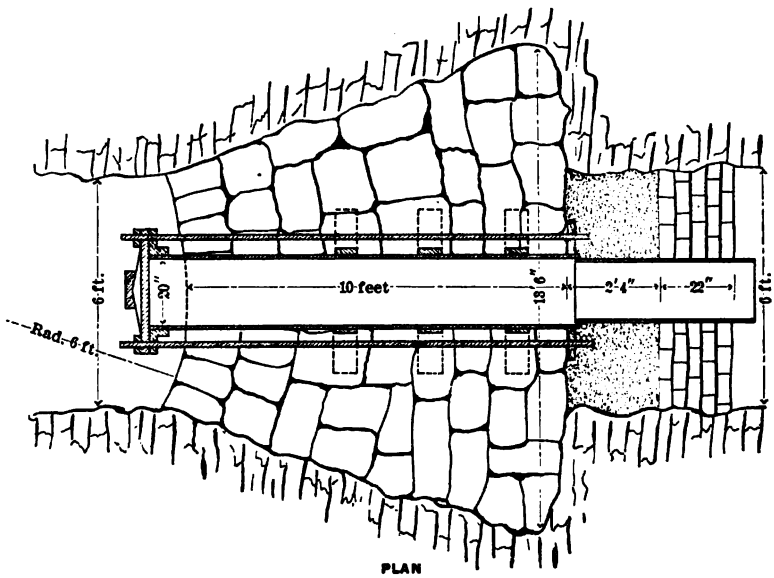
AN exploring-drift on the bottom level of the Curry mine, at Norway, Mich., on the Menominee range, cut a stream of water, which increased considerably the expense of pumping. The hope that the supply would drain off was not realized. It was decided to build a dam in the exploring-drift, as it was estimated that the cost would be recovered by the saving in fuel in less than two months.

The 8th and lowest level of the mine is 780 feet below the surface. The exploring-work had started north from the shaft at right angles across a slate to a jasper formation parallel to that which contains the principal ore-body. In this north jasper several hundred feet of drifts had been opened in different directions. The dam was located in the slate near the jasper. The drift at this point is 6 feet wide and 7½ feet high. The amount of water enclosed by the dam would be less if it had been built nearer the face of the openings; but the jasper, though hard to drill, is not firm, while the slates, though softer, are dense and solid, and the impermeability and strength of the adjoining measures were of commanding importance.

The dam was built of local sandstone in the shape of an arch upon its side, with a radius of 6 feet and a thickness of 10 feet. The mortar was made of one part Hilton cement and two parts sharp sand. The abutments were formed by cutting out the sides of the drift in line with the center of the curve of the arch, but leaving the rock rough. The floor was excavated 15 to 20 inches below the bottom of the drift, and the top was made 2 feet higher in front and 5 feet higher at the back. In laying the masonry the front courses of stone were cut to fit the arch and care was taken, particularly at the top, to leave no crevices. Two openings were left through the dam: one, a 5-inch pipe to carry off the water, fitted with a gate-valve at the outer end; and the other, a man-way of 20-inch steel pipe, plain at the inside end, and with a heavy flange shrunk on at the other, to which a heavily ribbed blind flange, 2½ inches thick, was bolted. A small pipe tapped into the blind flange carried a hydraulic pressure-gauge. The 20-inch pipe was anchored in the masonry by three clamps, and the flanges were bolted to three long rods passing through the wall with washers inside. The 5-inch pipe, with a flange on the inside end, was also anchored by three clamps, the arms of which radiated in different directions. As it is nearly 800 feet below the surface, and the source of the water was unknown, the pipe and fittings were designed to withstand a pressure of 350 pounds to the square inch.

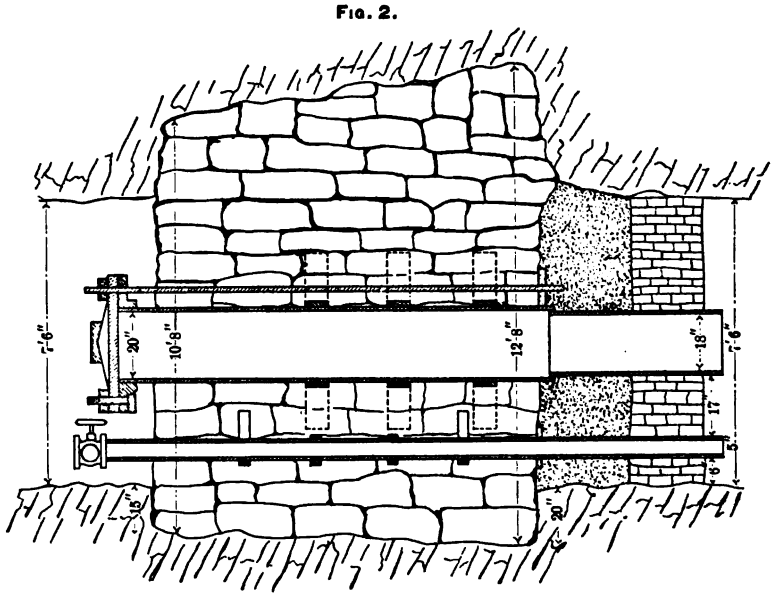
The water was shut off about 4.30 P.M., May 13, 1897, and a little before noon the next day the pressure was 100 pounds, rising to 215 pounds at 3.30 and 240 pounds three-quarters of an hour later. The dam leaked about 30 gallons a minute, the

FIG. 1.



PLAN

FIG. 2.



LONGITUDINAL SECTION

DAM IN CURRY MINE

Scale 1 inch = 5 feet.

quantity increasing with the pressure. This condition was not satisfactory, and the water was let off. In two hours the pressure fell to 45 pounds, and the water rose at the shaft about a foot over the floor. A high pressure was produced by a small quantity of water, showing that part of the water-course was very small.

The closing of the dam seems to have increased the flow. For the thirty days previous, the record of the pumping, measured by displacements, gave an average of 370 gallons per minute. After opening the dam, the average for fourteen days was 418 gallons.

To put in an impervious stratum, a brick wall 22 inches thick was built 2 feet 2 inches from the inside of the dam and the space between was filled with concrete. This was made by mixing four parts of limestone, broken to egg-size, with three parts of cement-mortar. A short length was added on the inside to the 5-inch pipe, and a piece of old smoke stack, 18 inches in diameter, was used for lining the extension of the man-way. Owing to the increase in the water after the dam was opened, there was some delay in draining the mine, and it was seventeen days before the addition to the dam was completed. Before closing it, three loads of horse-manure were put against the brick work and held there by a plank partition.

The water was shut off the second time on June 1st, at 9.30 A.M. While the water was accumulating in the drifts, the pressure increased slowly. The following are the readings of the gauge taken every two hours, beginning eight hours after the dam was closed:

Date.	Time.	Pressure, lbs. per sq. ft.	Equivalent head of water. Feet.	Increase in 2 hours. Feet.
June 1,	5.30 P.M.	5	12	
"	7.30 "	13	30	18
"	9.30 "	22	50	20
"	11.30 "	30	70	20
June 2,	1.30 A.M.	45	105	35
"	3.30 "	71	165	60
"	5.30 "	130	300	135
"	7.30 "	220	510	210
"	9.30 "	260	600	90
"	11.30 "	268	620	20
"	1.30 P.M.	270	622	2
"	3.30 "	271	625	3
"	5.30 "	272	628	3
"	7.30 "	273	630	2

On June 8th the pressure was 277 pounds to the square inch, equal to a head of 640 feet, and the total pressure against the dam was over 800 tons.

During the morning of June 2d, before the pressure had reached the maximum, the water which flowed from the dam was collected in a small stream and caught in a pail. It took 22 seconds to fill the pail, which held a little over 19 pints, showing a flow of about 7 gallons per minute.

The second day after, though the pressure was higher, the flow had decreased to 5 gallons, and four days later it had fallen to 2½ gallons per minute. The water which came through the dam trickled out in eight or ten different places, most of them near the top or one side. The water was milky and left a heavy calcareous deposit on the face of the dam and in the ditch.

After the dam was closed, the record of the pumps for six days showed that the mine was making 115 gallons of water per minute, a decrease of 295 gallons.

The cost of the dam was as follows:

For stone, brick and sand at top of shaft,	$ 41.25
" cement,	151.70
" pipes, valves and gauge,	95.00
" labor sending down materials,	16.54
" " breaking stone for concrete,	17.77
" " cutting out drift,	21.00
" " building dam,	141.01
Total,	$484.27

The stone was taken from an old foundation, and the cost of quarrying is not included.

The Potsdam* Gold-Ores of the Black Hills of South Dakota.

BY FRANK CLEMES SMITH, RAPID CITY, SOUTH DAKOTA.

(Lake Superior Meeting, July, 1897.)

In describing a certain class of ores, of ever-increasing importance in the Black Hills, they are variously called "refractory," "siliceous," or "Potsdam;" the first name serving to distinguish them from the other important class—the free mill-

* See Postscript, p. 428.

ing ores; the second name being obviously applied on account of their mineral character, and the third name being taken from their more frequent geological habitat—the Potsdam series of the Cambrian. For various reasons I have used the latter name in heading this article, and have further classed these ores as gold ores, when, to speak strictly, it might be unadvisable to adopt a classification based upon either geological habitat or economic character; as a matter of fact, neither qualifying adjective is strictly correct in universal application to the ores in question, since they are, in certain cases, found in rocks later than the Potsdam, and, in a few notable cases, are ores of silver. However, the above designation may be allowed to pass, since it will serve as a *quasi* distinction from the more general refractory and quartzose ores, and since these ores have been most largely mined in the Potsdam series; the further designation, as gold ores, may be allowable since, in the light of present knowledge, such are largely in the majority.

Paleozoic rocks, consisting of a maximum thickness of about 450 feet of Upper Cambrian or Potsdam (consisting of quartzites, sandstones and shales) conformably overlain by a maximum thickness of Carboniferous rocks of about 785 feet (mainly magnesian limestones, with some sandstones)* completely encircle the inner uplift of the Black Hills, and, at various points throughout the inner circle of the hills, varying thicknesses of Paleozoic rocks yet remain resting upon detached elevations of the Archean† measures.

The Potsdam quartzites, sandstones and shales, resulting from the degradation and subsequent littoral deposition of the subjacent Archean schists and slates, usually carry small amounts of gold, but neither these rocks nor those of the Carboniferous are ever sufficiently mineralized with the precious metals to be called ores, except in certain localities where they are cut by dikes, covered by flows or intercalated with sheets of igneous rocks. If this statement may be accepted, it will

* Dr. F. R. Carpenter, *Rept. Dak. School of Mines*, page 34, mentions the occurrence of Trenton limestone near Deadwood. Certain thin, blackish and lead-colored shales have been found in a few localities, and are possibly Devonian.

† The term Archean is used to cover all pre-Cambrian rocks exposed in the Black Hills, although they have been largely classed as Algonkian.

be at once understood, by those who are familiar with the more salient features of the geology of the Black Hills, that the districts capable of yielding siliceous or Potsdam ores are not necessarily to be limited to those already in existence, but are likely to be increased in number as the prospector visits new areas showing results of former igneous action.

At present, we may, for convenience, classify the producing area of the Black Hills into two districts: (1) The Northern connected district, and (2) the Galena district. Under the first we may include various previously-named districts, such as Bald Mountain, Ruby Basin, Portland, Ragged Top, Squaw Creek, Carbonate, and certain more or less neighboring localities covering detached areas of siliceous ore, such as the deposits near Lead City and Deadwood. In the Galena district we include the producing area in the vicinity of the town of Galena, which is, by wagon-road, some twelve miles southeasterly from Deadwood.

It should be understood that this classification into two districts is purely arbitrary, and is not made with reference to any organized mining districts, but simply for convenience in description. Both districts are included in the northern Black Hills, in Lawrence county, South Dakota.

The accompanying map is not of sufficient scope to show the Galena district, having been compiled from data taken in the School of Mines survey of the past summer, which had for its object the preliminary examination of only that country included in the so-called Northern connected district; that being, with respect to its superficial extent and its production in precious metals, the region *par excellence* of the siliceous or Potsdam ores. This district, moreover, by reason of its connected history and continuous production, furnishes the more complete knowledge of the characteristic Potsdam ores, and will form the basis for this paper, although the Galena district will be considered in its turn.

The Northern Connected District.

Topography.—With reference to the topography, I shall commence at Terry's peak, which is one of the most prominent in the Black Hills, and is located in the immediate vicinity of what is now the most important part of the siliceous ore-region. This

peak rises 7215 feet above sea-level; to the northward, about 5400 feet, is the town of Portland (on the Burlington & Missouri River railway), which is about 800 feet lower in elevation than Terry's peak; here rises Green mountain to an elevation of about 6540 feet. To the eastward, Green mountain slopes to the point (279 feet below the summit) of its junction with Bald mountain, from which the latter rises to about 6553 feet above sea-level; it is 3700 feet east of the summit of Green mountain. The old town of Portland (on the Fremont, Elkhorn & Missouri Valley R. R.) is about 1500 feet northwesterly from the summit of Green mountain. Something over 2.7 miles northwesterly from Terry's peak rises Elk mountain to about 6234 feet above sea-level; 5600 feet northwest of Elk mountain rises Ragged Top, to about 6108 feet. Two miles eastward of Terry's peak is Sugar Loaf hill, the altitude of which was not determined. To the eastward of a line, forming the meridian of Terry's peak and Green mountain, rise a number of spring-creeks, flowing easterly and northerly into certain main creeks, which lead into the Belle Fourche river. Immediately south of Green mountain, on the northern slope of Terry, Nevada creek flows easterly, in the deep gulch of that name; south of Nevada, on the east slope of Terry, rises Fantail creek, flowing, in Fantail gulch, eastward; still to the southward rise the heads of Whitetail creek, which flows easterly, uniting with the waters from Nevada and Fantail gulches, and carrying them, northerly, into Whitewood creek and on to the Belle Fourche river.

West of the Terry meridian rise many creeks and dry gulches, radiating in a general way from the Terry uplift, and, in a subordinate manner, from lesser elevations, such as Elk mountain and Ragged Top, and leading into Spearfish creek, whose waters empty, by way of Redwater creek, into the Belle Fourche river. Thus has been indicated a local northerly and southerly elevation, culminating near the meridian of Terry, and forming an eastern and western water-shed. Of the creeks named, Squaw creek, rising about one mile northwest of Green mountain, empties into Spearfish creek in a distance of about 3.6 miles, and falls in that distance something like 1950 feet; the gorge of this creek is very deep and its sides are usually exceedingly precipitous. South of Squaw creek is Long Val-

ley; it carries very little water, and, for the greater part of its upper course, its slope is relatively slight. Still to the southward, Jackass creek (which carries no water) and Calamity creek (in which is a spring-creek) unite in a precipitous cañon and lead into Spearfish creek. Still to the southward, Holmes and Johnson gulches unite and lead into Spearfish creek; both of these are dry gulches; and an old wagon-road leads from Spearfish creek through Johnson's gulch, forming the only practicable wagon-route (from the mouth of Squaw creek south along Spearfish creek to the mouth of Annie creek) for the connection of Spearfish valley with the Terry plateau. South of the last-named, McKinley creek leads into Spearfish creek, rising south of Elk mountain and carrying little water. Southeast of the mouth of McKinley creek, Annie creek empties into Spearfish creek; this creek rises on the northern slope of Terry, and receives the water of Ross Spring creek, also the water of Lost Camp creek, which rises on the western slope of Terry. The lower portion of Annie creek, below the junction with Lost Camp creek, is confined within precipitous banks, forming near the mouth a very narrow cañon. The other gulches to the southward (Sweet Betsy and Raspberry) are dry, and lead into Spearfish creek.

Geology.—In considering the geology of the Northern Connected district, a preface, with relation to certain general geological features of the Black Hills of South Dakota, is necessary. The general shape of the Black Hills uplift is that of an ellipse, whose major axis (some 80 miles in length) corresponds with the meridian of longitude 103° 45′ W., and whose minor axis (some 50 miles in length) corresponds with the parallel of latitude N. 44°. Roughly speaking, the eastern half of this ellipse is an area of Archean slates and schists, showing granitic exposures in its southern portion, and isolated peaks of phonolite, trachyte and quartz-porphyry in its northern portion. The western half of the ellipse is covered with limestone of Carboniferous age. Within the ellipse the limestone has a slight dip, while all around the margin of the ellipse it dips sharply away, forming an annular monocline or terrace. As already observed, the Paleozoic rocks are represented in the Black Hills by varying thicknesses of Upper Cambrian or Potsdam rocks and those of Carboniferous age, reaching a

maximum of 450 feet in the first case, and 785 feet in the second; Mesozoic rocks overlie the Carboniferous outside the margin of the ellipse, gradually increasing in thickness as they become more remote.

The geological history of the Black Hills may be summarized as follows: In Archean time, the Black Hills were already an island; the sea eroded the Archean rocks, and formed the sediments now represented by the Potsdam, doubtless almost entirely covering the original island, the possible exception being that of the Harney area in the southern hills. The next chapter in the stratigraphic history of the hills, wherein but an inconsiderable sedimentation of Silurian and Devonian rocks occurred, is supposed (1) to indicate a subsidence of the waters of the sea, whereby the Potsdam rocks became dry land, "with no displacements nor deep-cutting erosion," or (2) with especial reference to the fact of the universal conformability of the Potsdam and overlying Carboniferous rocks, to indicate a profound subsidence of the Black Hills area to abysmal depths, whereby, during Silurian and Devonian time, no (or but little) sedimentation occurred. The first theory above-mentioned continues the history by suggesting the rise of a new sea, depositing the Carboniferous limestones and, becoming shallower in the latter part of the age, depositing many beds of sandstones. According to the other theory, the sea-bottom was gradually raised, permitting the deposition of Carboniferous limestones, and later, as the sea-bottom came nearer the surface, of the sandstones. The change of level, through which occurred a gradual elevation in and east of the Rockies, was the result of orographic movements affecting the western margin of the Paleozoic sea; in this elevation the Black Hills participated, but its local elevation was subsequent. During the earlier Mesozoic age there is evidence that the Black Hills area could have been covered by but shallow seas, and that the final uplift occurred late in the Mesozoic age.

The results of igneous phenomena in the Black Hills afford a study of exceeding interest, which has yet received comparatively little attention. The occurrence of dikes in the vicinity of the eruptive peaks seems to have been generally overlooked; and little has been written concerning the character of the igneous rocks. The scope of this paper limits its consideration of

the igneous conditions to that of certain peaks within the limits of the accompanying map, and other igneous exposures in the same area, though the subject is worthy of much more extended consideration.

The peaks referred to are Terry's peak, Green and Bald mountains, Sugar Loaf, Elk mountain and Ragged Top. Of these, Terry's peak and Sugar Loaf are laccolites, and the others (excepting Bald mountain and Ragged Top, which require further study) are probably so. These peaks, together with others in the Black Hills of similar nature and rock constituents, have been variously classed, with regard to the geological period of their intrusion, as Jura-Triassic, post-Cretaceous and Tertiary. It has been further declared that they are all due to one period of eruptive activity, and that no evidence of volcanic lava-flows exists. If the expression "one period of eruptive activity" means one *period* or *epoch*, according to Dana's stratigraphic classification, the age of the igneous rocks* may be placed later than the White river beds of the Miocene Tertiary (pebbles of igneous rocks have also been found above the Dakota Cretaceous, near Rapid City). If the expression indicates a *continuous* period of eruptive activity, it is open to criticism, since there is evidence of at least two periods, as will be suggested later. With reference to igneous extrusives, I may be permitted to refer to a piece of obsidian taken from a mass forming what is known as McMillan's peak, near Brownsville, Lawrence county.

The apex of Terry's peak consists of basic quartz-porphyry; large deposits of talus of the same nature are distributed on its various slopes, with a few outcrops of rock in place to the northwestward. Intrusive sheets of the same rock occur, as shown in the following section of the Snowstorm mine-shaft, in Nevada gulch, about 3400 feet northerly from the apex of Terry, and about midway between that point and the apex of Green mountain.

	Feet.
(1) Porphyry,	124
(2) Shale,	10
(3) Porphyry (?),	85
(4) Shale,	30
(5) Porphyry,	4.5

* H. Newton, Geology and Resources of the Black Hills, p. 220.

	Feet.
(6) Shale,	87
(7) Porphyry,	0.5
(8) Lime Shale,	15
(9) Sand Rock,	12
(10) Quartzite,	6

This gives a total depth of 374 feet, of which 214 feet consists of igneous rock. No. 1, the only sample in the section which I have been able to examine, is unquestionably quartz-porphyry. Of the other igneous rocks cut in the shaft, No. 3 was called by the manager "porphyry or trachyte," Nos. 5 and 7 being called porphyry similar to No 1. Between this shaft and Terry's peak yet remain the upper beds of the Potsdam (above those cut in the shaft) to a thickness of possibly 275 feet, thus indicating pronounced laccolitic conditions for Terry's peak.

In the Potsdam areas about Terry's peak, vertical dikes of quartz-porphyry and rhyolite are frequently visible. On the western slope of Terry's peak, in the valley at the junction of Annie and Rose Spring creek, a low, jutting outcrop of coarsely crystalline phonolite of bluish color is visible; further down Annie creek, four distinct sheets of phonolite (apparently connected with the last-mentioned outcrop, though generally finer-grained) are intercalated with the Potsdam shales, and aggregate about 165 feet in thickness; an apparent dike of phonolite (similar to the first-mentioned variety) cuts through the horizontal Potsdam shales, near the point of junction of Annie with Lost Camp creek; still further down Annie creek, not far above its junction with Spearfish, the sides of the precipitous gorge (about 100 feet deep) are composed of a fine-grained leucitophyre which, some 40 feet above, is again exposed in the railroad cut. On the northwestern slope of Terry, Foley's flat is separated from Annie creek by a ridge terminating in Foley's point; on this ridge an outcrop of an exceedingly fetid quartz-porphyry was found. The summit of Green mountain shows a blocky outcrop of leucite-phonolite. Going down the eastern slope, to the valley separating Green from Bald mountain, various dikes of quartz-porphyry are seen, and *talus* of the same rock can be traced up the slope of Bald mountain for a distance of some 200 feet; here is visible a breccia of black slate (from the Archean underlying the Potsdam), quartz and porphyry, lying

in or near a quartz-porphyry dike; at the summit (about 100 feet farther up) a beautiful, pale-greenish quartz-porphyry is exposed. On the eastern side, about 250 feet below the summit, occurs an outcrop of phonolite. Elk mountain consists of an outcrop of basic quartz-porphyry similar to that of Terry, which stretches some distance in a westerly direction; its sides show considerable *talus*. About midway between Elk mountain and Spearfish creek, on what is known as Holmes's point, a thick sheet of leucite-phonolite covers a considerable area. On the eastern slope of Spearfish creek, nearly west of the last-named point, is another outcrop of leucite-phonolite. Ragged Top consists of a ragged ridge of trachytoid phonolite, stretching westerly. Along the ridge, east of Long valley, proceeding to Squaw creek by way of the Redpath trail, various outcrops of the igneous rocks are encountered, among which are quartz-porphyry, trachyte, mica-andesite, and phonolite. In the bed of Squaw creek, below the Redpath trail, is seen a thick sheet of trachyte underlying a portion of the Potsdam. About one-half mile further up this creek an outcrop of grayish phonolite may be seen, which is unique in that it shows a most marked slaty structure. From this point up to the old Portland mill, on the east fork of Squaw creek, a continual recurrence of igneous rocks may be seen, including most of the varieties mentioned. Southeasterly from Terry's peak, Sugar Loaf consists of a precipitous outcrop of trachytoid phonolite, from which a very thick sheet is seen in the bottom of White Tail gulch, underlying the upper workings of the Ross-Hannibal and other mines. At White Tail crossing, some distance easterly, quartz-porphyry dikes may be seen inter-bedded with the almost vertical Archean schists, between which occurs a dike of grayish-green phonolite; the quartz-porphyry contains fragments of the schist.

So much for the more important igneous phenomena characterizing and surrounding the peaks mentioned. In the map attached to this paper it is almost impossible to show the igneous rocks clearly; small outcrops and loose float are of almost universal occurrence over the entire Potsdam area. It will be noticed that quartz-porphyry or rhyolite and phonolite seem to be the prevailing rocks, and that these occur in sheets and dikes, as well as forming the summits of peaks. While these

and other igneous rocks of this district may occur as the result of one long-continued period of intermittent igneous activity, there would seem to be sufficient difference between extreme members of this rock-series (with regard to their acidity, as affecting their fusibility) to indicate the probability of the existence of wide intervals of time between their ejections. Thus we have rhyolite with from 75 to 82 per cent. of silica, quartz-porphyry with from 69 to 81 per cent., trachyte with from 55 to 62 per cent., phonolite with from 50 to 62 per cent., and leucitophyre with from 45 to 54 per cent. The relations of the dikes mentioned as existing at White Tail crossing may also be of moment in this consideration; and an additional proof is found at Bryant's mine, on the west slope of Squaw creek, where mica-andesite sheets are intercalated with horizontal layers of the Potsdam, and the whole is cut by vertical dikes of an igneous rock which is apparently trachyte, and which carries notable values in gold and silver, in the form of sylvanite, of which bright crystals are distinctly visible.

Entering this district along White Tail creek and proceeding westward, the valley consists of Archean rock; so also do the eastern extremities of the various outliers from Terry; but near the top of these outliers, the lower members of the Potsdam are seen outcropping across them, and dipping at a small angle toward Terry, or westward. Some distance further west the gulches are crossed by the Potsdam outcrops. Proceeding toward Terry, along the ridge between Nevada and Fantail gulches, to about the altitude of the town of Portland, the whole Potsdam series is crossed, showing the various quartzites, sandstones and shales, heavily stained with iron and cut by quartz-porphyry dikes. To the north, Archean rocks encircle the base of Bald and Green mountains, and the Potsdam (greatly cut by igneous rock) forms the surface as far north as Squaw creek. This Potsdam area (continually cut by dikes and intercalated with sheets of igneous rocks) extends westward to Redpath creek, thence its western limit follows the ridge east of Long valley toward U. S. Location Monument No. 60, from here running south to Annie creek, then along the eastern bank of Annie creek and along Lost Camp creek to a point about 2000 feet south of Terry's peak. Westward of this boundary line the Potsdam is covered with Carboniferous limestone (extending to Spearfish creek and beyond), and is only

visible in the various gulches leading into Spearfish creek and along the bed of that creek. To the northward of Squaw creek the country is again covered with Carboniferous limestone.

From the top of the Potsdam, on the ridge near the town of Portland, the Potsdam rocks are seen to dip westerly at an angle of about 12°, and further westward the conformable Carboniferous rocks have a similar dip. The extreme thickness of the Potsdam rocks of this district is about 450 feet; the Carboniferous limestone, first showing in a thin stratum along the western edge of the Potsdam, reaches a thickness at Spearfish creek of 450 feet or more, and, in the northern areas, a much greater thickness. Southeast of Terry, the phonolite flow from Sugar Loaf is widely extended and of great thickness.

Of the peaks mentioned, Terry and Elk mountain are outcrops of quartz-porphyry which have visibly cut through the Carboniferous and are at least younger than the rocks of that age; Elk mountain lies in a direction N. 55° W. from Terry's peak. Sugar Loaf, Green mountain and Ragged Top are outcrops of phonolite, of which the latter has visibly cut through the Carboniferous limestone and tilted it to a high angle on its western slope. These peaks lie in a general direction of N. 58° W. A line drawn through the quartz-porphyry peaks mentioned above shows a sort of parallelism to one drawn through the phonolite peaks, a fact which may have some significance. Bald mountain shows on its summit quartz-porphyry and, a short distance below, phonolite.

A noteworthy fact concerning these peaks, with the possible exception of Sugar Loaf and Ragged Top, is that in each case the tilting and disturbance of the sedimentary beds through which they have risen is remarkably local. All around Terry the upper portion of the Potsdam can be traced, and the same condition is observable at Bald and Green mountains, while, but a short distance away from either peak, the sedimentary rocks preserve their usual dips. Local faulting is prevalent and co-existent with most of the dike-phenomena, but is rarely excessive or noteworthy in amount.

Character, Occurrence and Origin of the Ores.—With regard to the character of the ores of this district, for convenience, then, two classes will be made, as follows:

(1) Gold Ores: (*a*) in the Potsdam,
(*b*) in the Carboniferous.
(2) Silver Ores: (*a*) in the Potsdam,
(*b*) in the Carboniferous.

(1*a*). The gold-ores of the Potsdam constitute a group of relatively much greater importance than all of the other siliceous ores together. While some deposits (such as that of the Dividend mine on Green mountain) yield pay-ore of a gouge-like decomposed character, in general the ores may be described as thoroughly reorganized sandstones, showing under the microscope many druses lined with innumerable quartz crystals, and containing calcite and fluorite. Of these ores, those which have not suffered oxidation frequently show considerable fine-grained pyrite and are locally called "blue" ores; the oxidized ores, though frequently showing only a small iron-content, are usually heavily stained with iron, and are called "red" ores. Both kinds are usually exceedingly tough and difficult to break or pulverize. Analyses* of typical samples yielded the following results:

Red Ore.

	Per cent.
Silica,	84.45
Alumina,	4.07
Ferric oxide,	7.28
Calcium oxide,	0.85
Magnesium oxide,	0.25
Sulphur trioxide,	3.71
Tellurium, 8.426 oz. per ton.	
Gold, 0.574 " "	
Silver, 2.875 " "	
Total,	100.61

Blue Ore.

	Per cent.
Silica,	68.748
Alumina,	3.072
Iron,	13.289
Sulphur,	11.728
Gypsum,	0.833
Fluorite,	0.784
Phosphorus pentoxide,	0.842
Tellurium, 4.03 oz. per ton.	
Gold, 0.325 " "	
Silver, 10.55 " "	
Total,	99.296

These may be considered as low-grade ores, and it is interesting to note that the analysis of the red ore might be almost

* See "The Occurrence and Behavior of Tellurium in Gold-Ores," *Trans.*, xxvi., 485.

duplicated by that of the blue ore after oxidation, during which the latter might be expected to lose iron and sulphur.

Taking averages of the tellurium, gold and silver found in the analysis of nine different samples of Potsdam ore, six being "red" and three being "blue," the following percentages were obtained:

	Per cent.
Tellurium,	59.97
Gold,	7.64
Silver,	32.39
Total,	100.00

According to Dr. Carpenter, silica in these ores averages about 78 per cent. Copper is usually found in but traces; analysis of a sample of precipitated gold from one of the chlorination-mills yielded copper to the amount of 10 per cent. of the total gold found. In the ores of Ruby basin the last-mentioned authority states that the copper and gold have been observed to occur in about equal amounts. Arsenic and antimony have been reported, but I have never been able to identify them.

The occurrence of these ores in this district is as eccentric and as widespread as is the igneous action; with respect to a section of the Potsdam formation the ores may be said to exist wherever mineralizing solutions permeated susceptible beds. In general, there are said to be two ore-bearing horizons or "contacts" (the upper existing relatively nearer the upper layer of the formation, and the lower found in the bottom quartzite); but at the Welcome mine, at the head of Fantail gulch, there are more than two, which may, however, be the result of local faulting. The ores of the lower contact are said to carry a greater proportion of gold to silver than those of the upper; but this is doubtful, if applied generally.

In considering the origin of these ores, the mineralization is undoubtedly referable to the igneous action which is so strongly marked in their vicinity. Throughout the district quartz-porphyry and rhyolite dikes are of universal occurrence, but seem to reach the surface less frequently in that part of the district which is covered by any considerable thickness of limestone. In fact, certain dikes have been observed to cease before reaching the upper horizon of the Potsdam. Wherever mineralization of the Potsdam beds has occurred, it can almost always be traced to a quartz-porphyry or rhyolite dike or "vertical," which itself is usually mineralized, stained with oxide of iron, and so much

broken and decomposed, that its rock-character is distinguishable with difficulty. In the immediate vicinity of this dike may exist another, consisting of the same igneous rock, but in a fresh, unbroken condition, and showing no evidence of having assisted in the phenomena of mineralization. The fact that one quartz-porphyry or rhyolite should, under similar conditions, be in every way as refractory as another, leads to the conclusion (1), that secondary disturbances have caused a selective shattering of certain dikes in preference to others (thus affording channels for the upward flow of mineralizing solutions), or, (2), that there has been a recurrence of dike-injections, coeval with which has occurred the necessary shattering of former dikes. That the mineralization is referable to secondary igneous action, whereby the lime-cement of the grains of certain Potsdam rocks has been replaced by siliceous paste carrying pyrite, sylvanite, a very small amount of copper, and rarely uranium minerals, is indicated by the character of the above minerals, and by the intimate relation of the ore-chutes to the igneous dikes, whereby the former are limited in length by the length of the shattered dike, in thickness by the thickness of the susceptible bed, and in width by the volume and strength of the mineralizing solutions. Much fluorite is also frequently found in these ores.

(1*b*). During the past year gold-ores have been found in the Carboniferous limestone; more particularly in the Dacy and neighboring mines, a short distance north of Ragged Top and west of the towns of Balmoral and Preston. Over the greater portion of the limestone area west, northwest and northerly from Terry's peak, the limestone is seen to be highly silicified in places, and frequently reddened with oxide of iron, and most of the croppings will at least give traces of gold upon assay. In the vicinity of the Dacy mine, gold was first discovered in boulders and pebbles, usually of a reddish color and resembling in most respects the ordinary siliceous ores of the Potsdam; further exploration developed wedge-like lenses of similar ores, frequently rich in gold and tightly encased in the limestone, together with loose boulders of the material. These deposits were at first supposed to pinch out within a few feet of the surface, but recent deeper working has proved that they can be traced deep into (if not quite through) the limestone, though in places they thin down to a mere streak. The upper workings

of the Dacy mine have yielded some ore which differed considerably in appearance from any of the siliceous ores hitherto found, being of the color and general appearance of the ordinary buff limestone, but evidently highly siliceous, and showing many small angular cavities. An analysis of a sample of this ore gave the following result:

	Per cent.
Moisture,	0.110
Volatile matter,	0.802
Silica.	90.990
Alumina,	2.970
Ferrous oxide,	3.024
Calcium oxide,	1.138
Magnesium oxide,	trace
Tellurium, 29.26 oz. per ton.	
Gold, 17.34 " "	
Silver, 1.21 " "	
Total,	99.034

Combining the gold, silver and tellurium in the above analysis, we find them existing in the following relative percentages:

	Per cent.
Tellurium,	61.20
Gold,	36.27
Silver,	2.53
Total,	100.00

This closely approximates the composition of sylvanite, according to Klaproth's analysis.

In the Dacy shaft, which is now at a depth of about 385 feet and is evidently nearly through the limestone, the crevice or fissure through which mineralization has occurred may be seen. In places it is still open, with no filling; again, it is filled with a breccia of silicified limestone fragments, cemented by crystalline calcite; wherever the walls are visible may be seen eminent siliceous incrustations rounding off the angular points of the fractured limestone. Irregularly scattered through the limestone are many bluish translucent flinty nodules of varying sizes; these, in some cases, are said to be rich in gold, though I have not yet found a sample which yielded over $12 in gold per ton upon assay. At another working upon the Dacy ground it has been found advisable to screen the ore coarsely as it comes to the surface. In this way a fine material, consisting largely of carbonate of lime, and carrying small value

in gold, is separated from the coarser and highly siliceous material which is shipped to the reduction-works.

In the light of the present development of the Dacy mine it is impossible to determine the shape of the ore-body at any considerable depth below the surface or to offer further description of it. It is, however, safe to say that the mineralized portion of the deposit is genetically connected with the fissure already described.

Many other workings of smaller extent surround the Dacy property, none of them showing differing conditions from those of the Dacy mine, as far as their development has progressed. Some miles to the northward, however, near the top of the high limestone cliff overlooking Spearfish creek, certain deposits occur in the limestone which resemble mineralogically and in their horizontal position the siliceous ores of the Potsdam; these deposits are known to carry some gold, and late workings are reported to have developed pay-ore.

The origin of the gold-ores of the Carboniferous seems to be undoubtedly traceable to the phenomena which caused the mineralization of the Potsdam; but, as I have previously said, in the limestone areas the igneous dikes are much less strongly in evidence. Recently my attention was called to an igneous dike, cutting the limestone in the mine-workings of Mr. John Doyle. The rock was much decomposed and carried slate inclusions.

(2*a*). As already observed, the ores of the upper contacts in the Potsdam are said to carry a relatively higher percentage of silver than those of the lower. This is occasionally true; and, in certain localities, the ores of the upper contact are of such a nature as to be typical silver-ores, such as the first ores mined in the Empire and Trojan mines on Green mountain, and a certain class of ores from the Ross-Hannibal mine, one sample of which was of a dark bluish color, showing crystals of torbernite and assaying 255.08 ounces of silver and 0.44 ounce of gold per ton. The Ross-Hannibal workings are located not far from the phonolitic Sugar Loaf and are underlain by a thick phonolite sheet.

(2*b*). In the mine-workings on Elk mountain (which is a quartz-porphyry uplift piercing the Carboniferous) silver-ores occur, one sample assaying 17.16 ounces of silver and 0.24 ounce of gold per ton. Away to the northward, in the Car-

bonate camp, in the Carboniferous limestone at the Iron Hill mine, a heavily iron-stained siliceous rock carries ores of silver; lead-ores are also present. Some samples of these silver-ores show large amounts of cerargyrite. In this mine a vertical porphyry dike is reported,* while between the Carboniferous and the Potsdam, as seen in the bed of Cold creek, is a horizontal sheet of the same rock, and the underlying Potsdam rocks are mineralized with gold.

The origin of these deposits of silver-ores is probably intimately connected with that of the gold-ores; in amount the silver-ores are relatively of very little importance, the mines producing them having not been lately worked to any great extent (except the Ross-Hannibal, which is producing gold-ores) and relatively little is known about them.

The foregoing observations with regard to the character and appearance of these ores would indicate the possibility of some such history as the following: First occurred a period during which the intrusion of Terry's peak and Elk mountain took place, with widespread dike- and flow-action of quartz-porphyry and rhyolite. During or immediately subsequent to this highly acid igneous action, little if any mineralization occurred. After an indeterminate period (during which lavas of varying fusibility may have been erupted, some notable fracturing of former dikes may have occurred, and also some mineralization) occurred the most basic eruption of phonolites (represented at Sugar Loaf, Green Mountain and Ragged Top and in numerous sheets) producing the final fracturing of former dikes, also fracturing the limestone (as at the Dacy mine, where the limestone seems too remote from Elk mountain to owe to that uplift its fractured condition) and causing the efflux of mineral-bearing waters which ascended through fractured dikes and metamorphosed portions of the Potsdam beds and, ascending in the limestone fissures, performed for the magnesian limestone the same function.

History.—The history of the Potsdam ores commences in 1877 when Mr. A. J. Smith, of Portland, South Dakota, located the Empire mine on Green mountain and later the Trojan, Perseverance and Indispensable in the same vicinity. In 1879 the first engine was erected to supply power for the treatment

* Dr. Carpenter, *Report Dak. School of Mines*, p. 126.

of these ores; with it an arrastra was run upon ores from the Empire mine averaging about $35 per ton. The saving in precious metals was little or nothing, and the attempt was discontinued after about two months. In 1880 the Portland Mining Company, owning the Portland, Gustavus and Pilgrim mining claims, built a mill (which still stands idle near the old town of Portland) for the treatment of these ores by pan-amalgamation; later, this company purchased the Empire, Trojan, Indispensable, Perseverance, Folger and Olive claims. Pan-amalgamation saved about 50 per cent. of the silver and 30 per cent. of the gold, and various other processes, such as "free-milling," kiln-roasting and chlorination, and cyanide lixiviation were tried without success. In 1883 the Welcome Mining Company passed through a similar history of unfortunate metallurgical experiment upon their property near the head of Fantail gulch. In 1886 the Buxton mill was built by the Buxton Mining Company, and experiments made there by Mr. O. P. Ankeny, by the use of bromine, seemed to have stopped just short of success, probably on account of disadvantages which, at the present date, would not surround the experiment. About this time the siliceous ores were successfully treated by the Plattner process in the metallurgical department of the School of Mines. In 1889 the Golden Reward Gold Mining Company erected a plant in Deadwood for the treatment of the Potsdam ores by barrel-chlorination, commencing in April, 1891, with a capacity of 50 tons per day. In 1890 the Deadwood and Delaware Smelting Company built the small experimental smelting-plant in Deadwood which has since grown so largely. Two years later, or in 1892, a small cyanide-plant was erected in Deadwood by the Black Hills Gold and Extraction Company; and in November, 1895, the Kildonan Milling Company started chlorination-works at Pluma, South Dakota (a short distance from Deadwood), with a capacity of about 75 tons per day.

During the time from the first disastrous attempts at the treatment of the Potsdam ores until more skillful experiment had firmly established the Deadwood and Delaware and the Golden Reward plants, although considerable amounts of ore were shipped outside of the Black Hills for treatment, the fact that ores of less value than about $30 per ton could not be mined and shipped with any profit, together with a superstitious

feeling that there was something "uncanny" about these ores and that they could not be treated successfully, prevented an active development of the Potsdam ore-bodies. Only within the last three years have the old conditions been changed; and miners can now generally obtain a profit from ores carrying $15 per ton. Naturally the earlier mines in the Potsdam ore-bodies were located where the Potsdam rocks were easiest of attack, in the vicinity of Green and Bald mountains and to the east of Terry's peak in Nevada, Fantail and Whitetail gulches. In these localities the three large companies which handle the Potsdam ores (the Deadwood and Delaware, Golden Reward and Horseshoe) own and operate a large number of very valuable mines, and smaller companies and individual owners hold the remainder. Mining is not expensive; a large amount of it being carried on by means of tunnels, and the shafts being of relatively small depth. Little timbering is necessary, but water, in some cases, causes some annoyance; the ores are usually very hard and tough; and the eccentricity of the ore-chutes, with local faulting, add to the expense of dead-work. The following section, from a drill-hole on Silver Reef claim, about one mile northeasterly from Terry's peak, will show the character of the Potsdam rocks:

	Ft.	In.
Rhyolite,	43	—
Hard and soft lime shales, "pudding" variety,	16	8
Same as above,	12	10
Dark mud shales; split readily on exposure,	10	—
Hard "pudding" lime shales,	8	—
Hard lime shales,	9	—
Alternate bands of hard lime shales and green lime shales, also traces of blue ore,	14	—
Hard blue clay shales,	7	—
Glauconitic shales with pyrite,	2	6
Blue clay shales and "pudding" lime shales,	14	—
Hard "pudding" lime shales. 8 inches of hard blue-black limestone, and 6 inches of hard "pudding" shales,	20	—
Uniform yellow clay. True cap of lower ore,	6	—
Potsdam sandstone; red and yellow on top. True ore-bearing rock,	17	10
Brilliant red gouge,	1	6
True quartzite,	19	—
Sand rock,	5	—
Cemented sand and quartz pebbles,	2	—
Archean slates,	8	8
	217	

Metallurgy of the Potsdam Ores.—Although many years ago the fire-assay of the Potsdam ores showed that they contained

gold and silver in notable quantities, it seems to be the fact that until very lately no sufficiently comprehensive chemical analyses were ever made to determine in what condition the precious metals existed. As already remarked, pyrite occurs plentifully in some of the blue ores; and doubtless that mineral was supposed to render them refractory, or, from the fact that only in very rare cases and by the closest examination can any free gold be detected, it may have been supposed that the gold existed in particles too minute to be saved by ordinary processes. Screen-analyses show that the precious metals are distributed throughout the ores with great uniformity, and chemical analyses of a large number of samples show that they are associated with tellurium. In addition to the early lack of metallurgical knowledge it would seem that too profuse litigation as well as the clashing of personal interests finally terminated the history of some of the early metallurgical plants, even after they had operated profitably, and greatly retarded the mining and metallurgical development of the Black Hills. Naming them in the order of their initiation, chlorination, matte-smelting and cyanide lixiviation have (operating almost exclusively upon Potsdam ores) developed into remarkable industries in the Black Hills, particularly the first two. Chlorination, as exemplified at the Golden Reward and Kildonan mills, is carried on in the following manner: The ore is first put through rock-breakers, from which it passes to a drier; from the drier it passes to fine crushers, and thence to a series of crushing-rolls, where it is crushed to 8- or 12-mesh. It is next roasted in Howell-White continuous-discharge cylinders in the case of red ores, or in Brückner cylinders in the case of blue ores. After roasting, the product is spread on cooling-floors, from which it is elevated to bins to be discharged by hoppers into the chlorination-barrels. The barrels are of iron, lead-lined, and have a capacity of about five tons. About four tons of ore are charged along with about twelve pounds of bleaching powder and from eighteen to twenty pounds of sulphuric acid per ton; the barrels are slowly revolved for about one and one-half hours, water being admitted under pressure after chlorination is complete, and leaching the chlorinated ore through a filter contained in the barrel. The gold-bearing solution is treated with hydro-sulphuric acid gas, in some cases

after a preliminary treatment with sulphurous acid to reduce chlorine. In this treatment gold, along with various amounts of other metals, is precipitated; the solution is filtered through filter-presses; the collected sulphides are dried, roasted in muffles and smelted to bullion. Any silver contained in the ore chlorinated is, of course, lost, and a discrimination is usually made with reference to the ores treated by this process, whereby all ores carrying over three ounces of silver per ton are reserved for other treatment. At the Golden Reward mill a cyanide-plant has been added for the treatment of this class of ores, making use of similar revolving barrels to those used in chlorination. At the same mill additional roasting-capacity has been secured by the erection of a Brown Horseshoe roasting-furnace, which is said to give great satisfaction. The Golden Reward mill has a maximum capacity of about 150 tons per day of red ores, or about 100 tons per day of blue ores. The blue ores are roasted in Brückner cylinders, while the red ores are in part treated in the Brown roaster, which has a capacity of 65 tons in 24 hours. The Kildonan mill has a capacity of about 100 tons per day, crushing to about 8-mesh. Neither mill treats any custom-ores, but each operates exclusively upon the product of its own mines.

While the efficiency of these mills is evidently satisfactory to their owners, their saving in precious metals is not so good as might be desired or as will be obtained in the future. Finer crushing would undoubtedly insure a better extraction, but would also add to the expense as well as interfere with filtration. It is to be supposed that those operating these mills have experimented along various possible lines with a view to secure the most economic method for the treatment of these ores, both with respect to a better saving of gold and the saving of at least a part of the silver; it is also to be supposed that, with the present arrangement of the respective plants, they are producing relatively good results. The Golden Reward has trebled its initial capacity, while that of the Kildonan has been nearly doubled during its existence of about one year.

From various points of view the Deadwood and Delaware Smelting Works constitutes the most important metallurgical industry in the Black Hills, and its existence is of a special value from the fact that it furnishes the only local market for Pots-

dam ores. Commencing in a comparatively small way, in its life of about seven years it has developed a daily capacity of about 400 tons, of which about 180 tons are Potsdam ores. The process, as developed by Dr. Carpenter, is one of matte-concentration, wherein the blast-furnaces are charged with the following burden:

	Tons.
Siliceous ore,	180
Copper-ore,	60
Pyrite,	100
Limestone,	180

This mixture is smelted with from 10 to 12 per cent. by weight of eastern coke or about 16 per cent. of Newcastle coke. Slags carrying 10 to 20 cents per ton in gold, 0.2 to 0.4 ounce of silver per ton and traces only of copper, are separated in the fore-hearth, and a matte is formed carrying all of the gold and 90 per cent. of the silver, as calculated from the assay of the ores charged, and constituting about 5 per cent. by weight of the original charge. The magnesian limestone is obtained from the Carboniferous outcrop, but a short distance from the smelter, at a cost of 50 cents per ton; the pyrite is obtained in the form of concentrates from the Homestake mine; copper-ores are brought from Montana and are added according to the supply, the charge containing from 2 to 10 per cent. of copper, with a probable average of about 5 per cent.

Matte-smelting, under the special form used in the plant just mentioned, indicates a great advance in metallurgical progress; as compared with the other processes mentioned, the saving of the precious metals by this process is much better. Compared with lead-smelting, for which either lead or copper must be brought in from beyond the hills, its efficiency is much greater, owing to the highly siliceous charges possible. In lead-smelting slags are calculated having an extreme upper limit in silica of about 37 per cent., while at the Deadwood and Delaware works the ordinary slag carries about 52 per cent. of silica. The charge for treatment of Potsdam ores at these works is $10 per ton, allowing 95 per cent. of the gold and paying for 90 per cent. of the silver over one ounce per ton.

The figures given above, indicating a daily treatment of 520 tons of material, represent the capacity of three blast-furnaces.

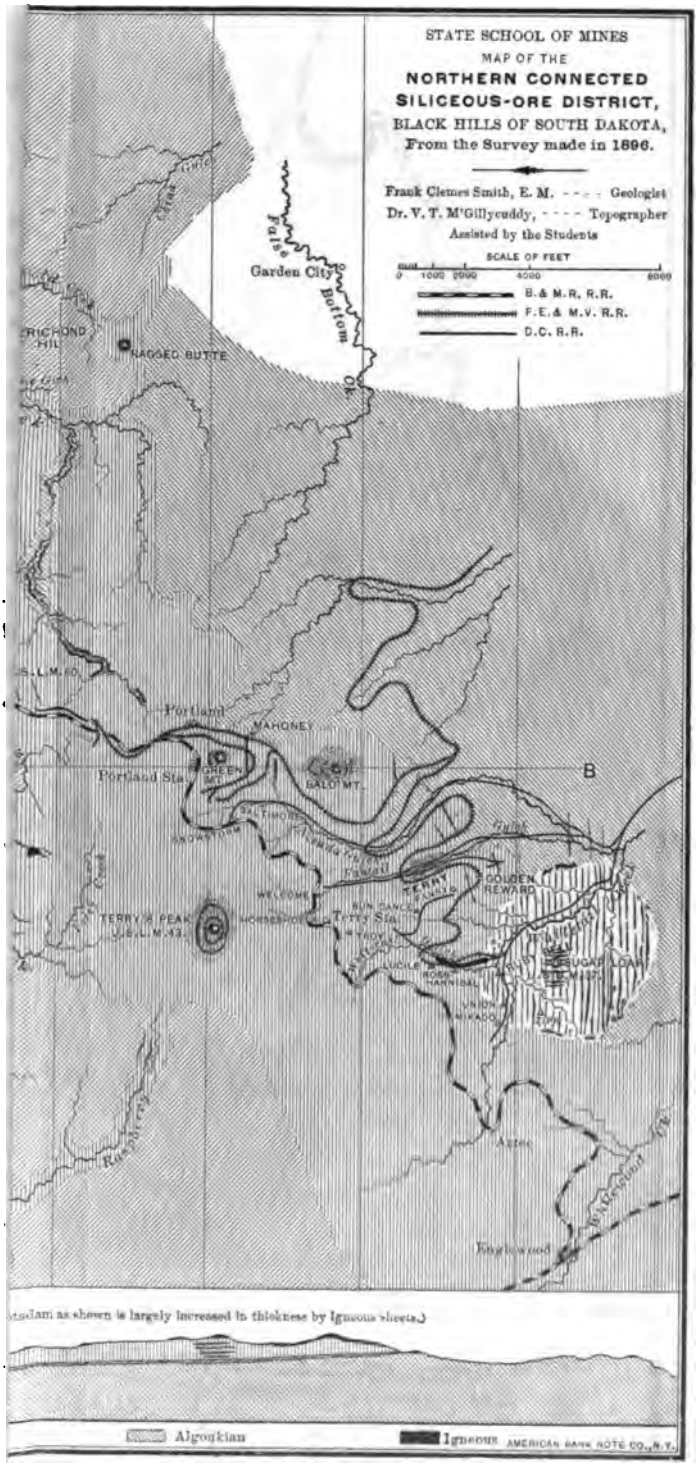
The plant consists of five, of which three are continuously in blast. There are also two reverberatory furnaces for the preliminary treatment or slag-roasting of fine ores and concentrates.

The cyanide-process is represented in the Black Hills by the plant of the Black Hills Gold and Extraction Company at Deadwood, which has a capacity of about 30 tons per day, but which has not been constantly in operation. In this plant, which seems to be rather badly arranged, only red or oxidized Potsdam ores are treated, being fed to the crushers along with sufficient lime to counteract the acidity; thence the ore goes to the drier, then through another crusher and three sets of rolls, with the intention of crushing to 30-mesh. The crushed ore is leached in tanks in the ordinary manner, the tailings being removed by hand and reserved for future treatment. The gold is precipitated from the solution by zinc-shavings, and the handling of the precipitate does not differ from the usual practice.

Present Output.—The tonnage of Potsdam ores treated is being constantly increased and the area in which these ores are mined is ever extending. The recent discovery of pay-ore at Ragged Top has caused a typical mining "boom," and will result in the development of a section of country in which, heretofore, little has been done in the way of mining. The smelter, with a view both to the treatment of the product of its own mines and to increased custom work, is continually enlarging its capacity; and this is also true, to a lesser degree, of the chlorination-mills. The following may be considered a conservative estimate of the tonnage of Potsdam ores treated during the year 1896:

	Tons.
D. & D. Smelting Co.,	65,700
Golden Reward Co. (Chlorination),	39,000
Kildonan Milling Co. (Chlorination),	36,000
Black Hills G. & E. Co. (Cyanide),	6,500
Shipped outside, about,	4,800
Total,	152,000

In the consideration of the Northern connected district but little mention has been made of the more or less disconnected areas of Potsdam ores in the neighborhood of Lead City and Deadwood, yet there are many valuable producing mines among


STATE SCHOOL OF MINES
MAP OF THE
NORTHERN CONNECTED
SILICEOUS-ORE DISTRICT,
BLACK HILLS OF SOUTH DAKOTA,
From the Survey made in 1896.
Frank Clemes Smith, E. M. - - - - Geologist
Dr. V. T. M'Gillycuddy, - - - - Topographer
Assisted by the Students
SCALE OF FEET
0 1000 2000 4000 8000
B. & M. R. R. R.
F. E. & M. V. R. R.
D. C. R. R.
Garden City
False Bottom Ck.
RAGGED BUTTE
Portland
MAHONEY
Portland Sta.
GREEN MT.
BALD MT.
B
GOLDEN REWARD
TERRY
WELCOME
TERRY'S PEAK
U.S.L.M.43
Terry Sta.
SUGAR LOAF
Aztec
Englewood
Algonkian
Igneous
AMERICAN BANK NOTE CO., N.Y.

them, and large tracts are yet undeveloped; the ores are similar to those already described.

The Galena District.

Fifteen years ago the town of Galena, situated some twelve miles southeasterly from Deadwood, could be called a lively and productive camp; the Davy smelting works was in operation upon ores from the Sitting Bull mine, and the yield is reported as little short of $750,000. A quartz mill was in operation and another smelting-works was erected, when suddenly all work stopped; and little has been done, except in the way of prospecting, until during the past year. The smelting-works has been remodelled and put in blast; the stamp-mill has also been put in running order, and various other preparations are reported, indicating renewed activity in this interesting camp. The principal producing mines are located in and near the town of Galena, where the Potsdam rocks are cut through to the underlying Archean; the ores carry silver and lead, and are found replacing calcareous shale-beds, and in seams and pockets in the quartzite which is itself, in places, highly charged with pyrite. Igneous dikes everywhere cut the sedimentary rocks, and the mineralization is evidently connected with their presence. The district, together with a large area adjoining, has not received its proper share of attention and study; there is little doubt, however, that this will follow the late evidences of a renewal of business activity.

In conclusion, reference should be made to another class of ores which may be as truly called Potsdam ores as those already described; these are the conglomerates, which are found everywhere between the base of the Potsdam and the Archean, and which, occasionally, carry valuable amounts of gold. In certain places, notably in the vicinity of the Homestake outcrop in Lawrence county and at Rockerville, Pennington county, these conglomerates have been worked for years. They are, however, simply placer-deposits, in which the gold is largely free, and are of secondary importance to the typical Potsdam ores, both in value and amount.

I take pleasure in expressing my gratitude to Dr. F. R. Carpenter for valuable information upon various points concerning the Potsdam ores and for kindly criticism of my work throughout this paper.

Postscript.

Since the first publication of the above paper, the writer has been informed by Director Charles D. Walcott, of the U. S. Geological Survey, that the Cambrian rocks of the Black Hills belong to the *middle*, not the *upper*, series: hence the name Potsdam has been wrongly used.

Notes on the Tin-Deposits of Mexico.

BY WALTER RENTON INGALLS, NEW YORK CITY.

(Lake Superior Meeting, July, 1897.)

In addition to the localities in which tin-ores have been found in Mexico, mentioned in my paper on the "Tin-Deposits of Durango," presented to the Institute in March, 1895,* the following should be recorded. I am indebted to Captain Henry Freeman, of Durango, for the notes as to these occurrences.

In the Sierra de la Candela, west of the Sierra de San Francisco and El Promontorio (the Llano de Guatimape intervening between the two ranges), cassiterite is found in veins and stream-beds. The veins are narrow, and the ores have the same characteristics as those of Potrillos. They were exploited by miners from Coneto in 1882–86, the more extensive work having been done on the northern side of the mountains.

In the Sierra de las Palomas, on the western edge of the Otinapa ranch, about 25 miles W.N.W. from the Diablo mine in the Cacária district, there are several small veins, of which the ore, gangue and country-rock are similar to those of the Cerro de los Remedios, near the city of Durango. The latter are essentially the same as the Potrillos veins.

In the Cerritos del Olote, in the Murphy grant, about 50 miles W.S.W. from Durango, there are old mines, now partially filled. The veins are narrow and poor; the ore is highly ferruginous; the country-rock is rhyolite, with some granite; immense quantities of chalcedonic rubbish are strewn along the course of the veins.

In the Cerro Blanco, overlooking the pueblo of Mezquital, 48 miles south of Durango, tin-ore is mined by the natives.

* *Trans.*, xxv., 146.

The mountains are of an eruptive character, rising 3000 feet above the valley of Mezquital, and sloping eastward and south-eastward to join the Sierra de Michis. The formation is granite and trachyte. Tin-ore is found in ill-defined veins in the decomposed trachyte. The veins are narrow (from 18 to 26 inches wide), but the ore is rich and free from arsenic and antimony, and is smelted easily in the Mexican adobe furnaces. The veins are worked in a desultory manner, between January and June, by the Indians of Mezquital, who refine the product with the aid of resin (*bréa*) and the green branches of the red oak, and sell it to the store-keepers of the pueblo at 20 cents (silver) per pound, in "trade."

At Guanajuatillo, a series of ridges in the Sierra de Michis, about 40 miles S.E. of Cerro Blanco, and 85 miles S.S.E. from Durango, there are several strong veins, in places 10 to 14 feet wide, of very ferruginous tin-ore. Captain Freeman says that he smelted 1300 pounds of this ore and obtained 13.1 per cent. of fairly clean, merchantable tin; but owing to the situation of the mines, the experiment was unprofitable.

In the Cordon de la Sierra de Sacrificio, 68 miles east of Durango, and 28 miles east of Nombre de Dios, on the main road to Zacatecas, stream- and vein-deposits were worked by *gambucinos*, previous to 1881. The stream-tin was dressed to 50–56 per cent. SnO_2, and the lode-tin to 30–37 per cent. The lode-tin is contaminated with iron, arsenic, and wolfram minerals.

In my former paper I referred to an analysis of metal from Canitas, Zacatecas, which probably came from Sain Alto in the same state, Canitas being the nearest railway point. The localities now mentioned by no means complete the list of tin-occurrences in Mexico. It is found at various places in Guanajuato, whence small lots of metal are from time to time brought into the towns by natives. Aside from the native smelting, the total of which is insignificant, there is no production of tin in Mexico at the present time. So far as information can be obtained, all the occurrences of tin-ore in the Republic are in young, highly acid igneous rocks.

A Combination Retort and Reverberatory Furnace.

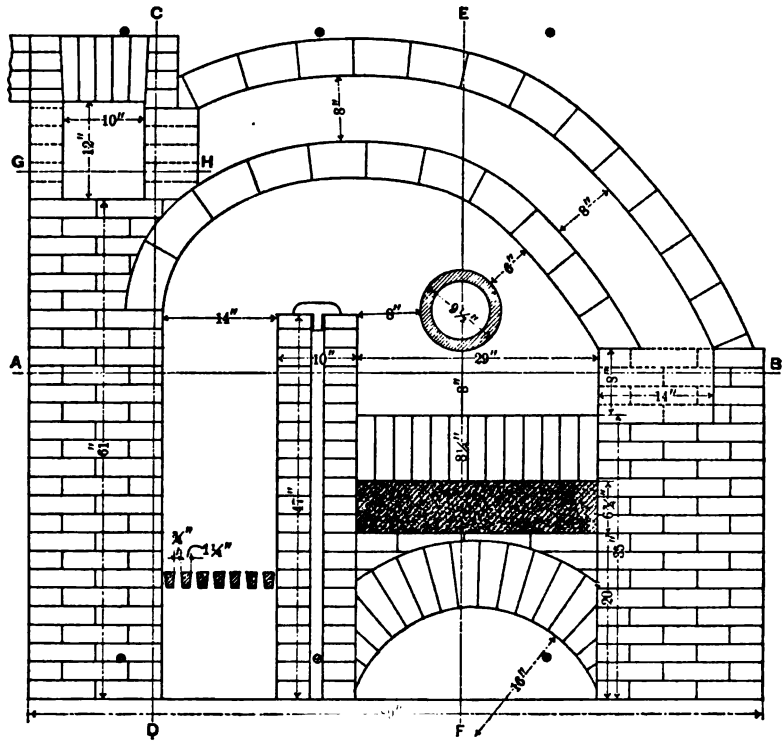
BY COURTENAY DE KALB, NEW YORK CITY.

(Lake Superior Meeting, July, 1897.)

THE accompanying illustrations, Figs. 1 to 6, from plans drawn by Mr. T. L. Knapp, a student in the Missouri School of Mines, show a type of furnace designed by the writer, which possesses some peculiarities intended to meet the necessities of laboratory-instruction in that institution. The object aimed at was the construction of a simple, inexpensive furnace, suitable for the distillation of zinc from its ores, and using only a single retort. This limitation imposed the chief difficulty, owing to the necessity of obtaining a high degree of heat concentrated upon the retort. This is accomplished by the peculiar curve of the arch, which consists of arcs of seven distinct ellipses, one focus of each of which lies within the area occupied by the retort. Thus the reflected heat would necessarily be thrown mainly where most needed, namely, on the retort. Conservation of heat is provided for by a return-flue through the hollow arch, and by a sand bottom under the fire-brick hearth. Four openings lead back from the laboratory, or heating-chamber in this instance, to the arch-flue; and three openings lead from the arch-flue to a large horizontal flue, from which a down-take conducts the gases to the stack. The fire-box, it will be observed, is deep, thus producing much CO during the combustion of the coal. The further combustion of this CO may be accomplished by the admission of air through the hollow fire-bridge, thus producing a higher flame-temperature surrounding the retort.

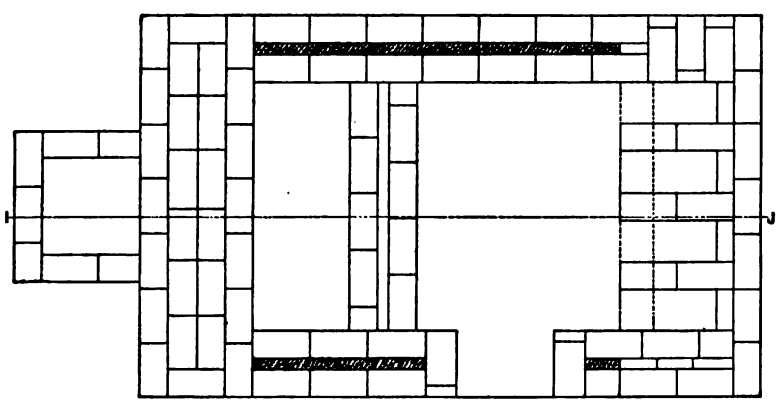
Provision is also made for removing the retort and the portion of the wall supporting it and the condenser, leaving a high arch, into which a door-frame may be inserted; the space above being temporarily bricked in. By this alteration, which may be quickly and easily made, the furnace may be converted into a reverberatory, suitable for treating small charges such as

FIG. 1.



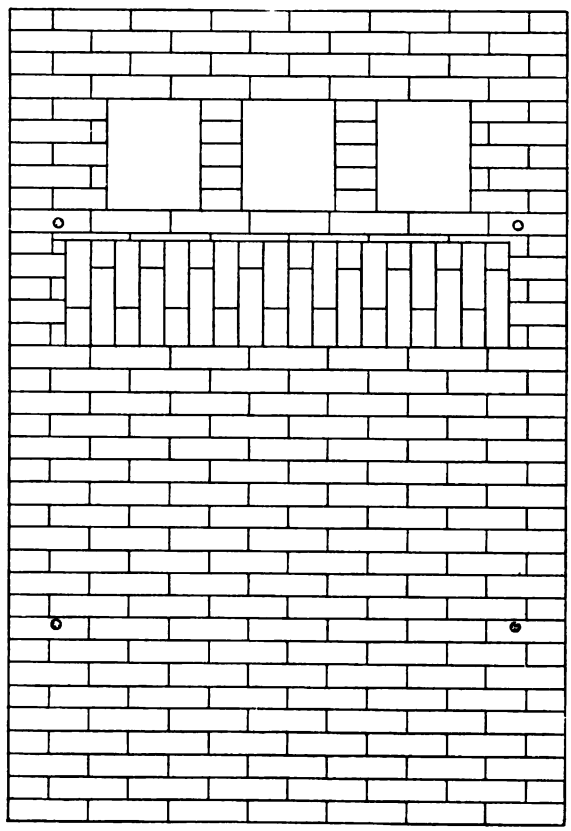
Section I J, Fig. 2.

FIG. 2.



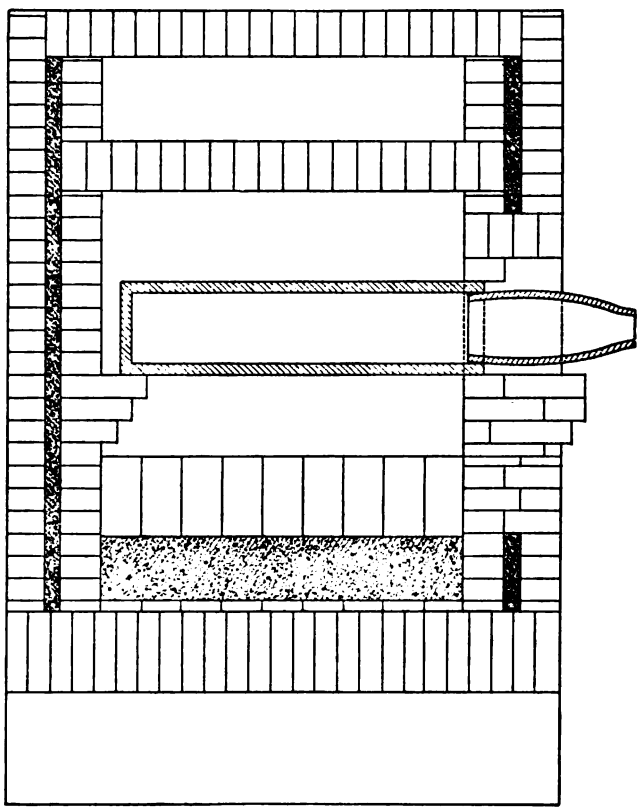
Section A B, Fig. 1, without Tying Rods.

FIG. 3.



Section C D, Fig. 1.

FIG. 4.



Section E F, FIG. 1.

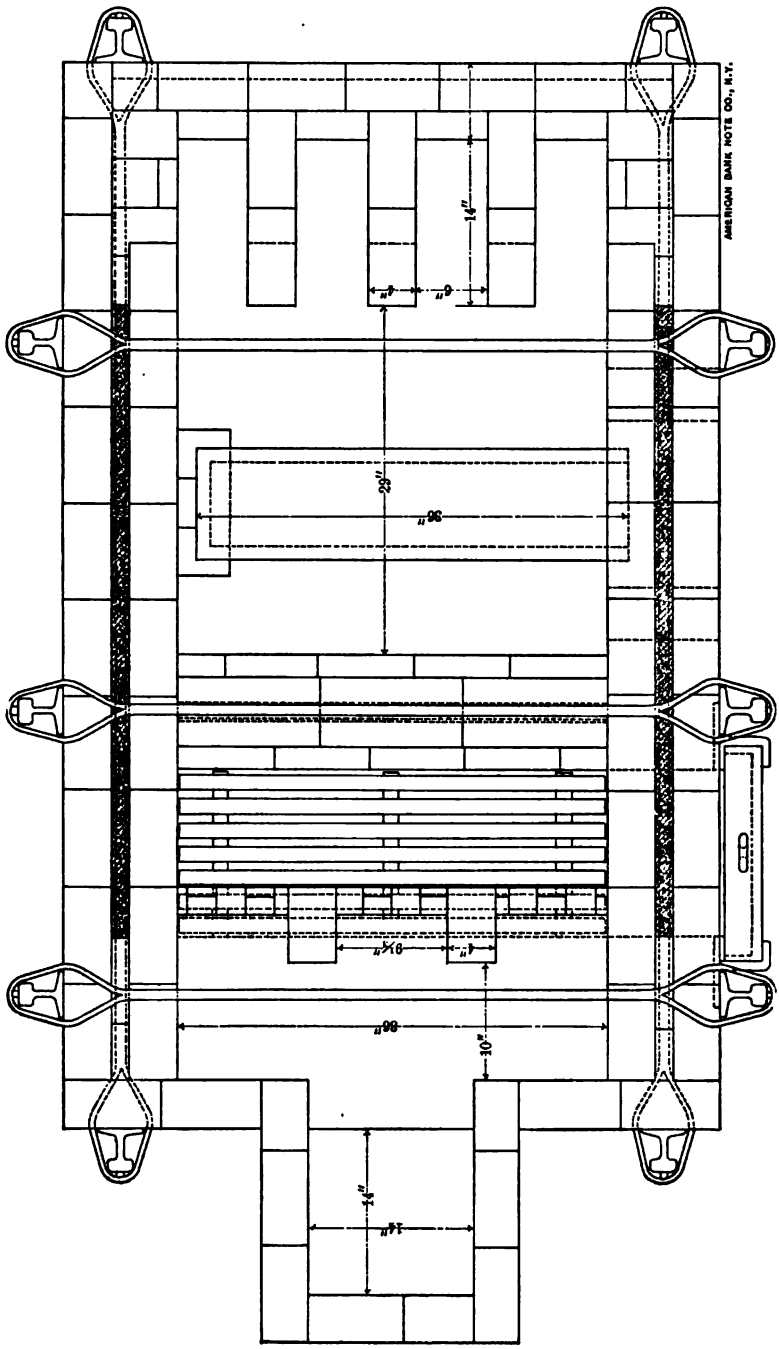
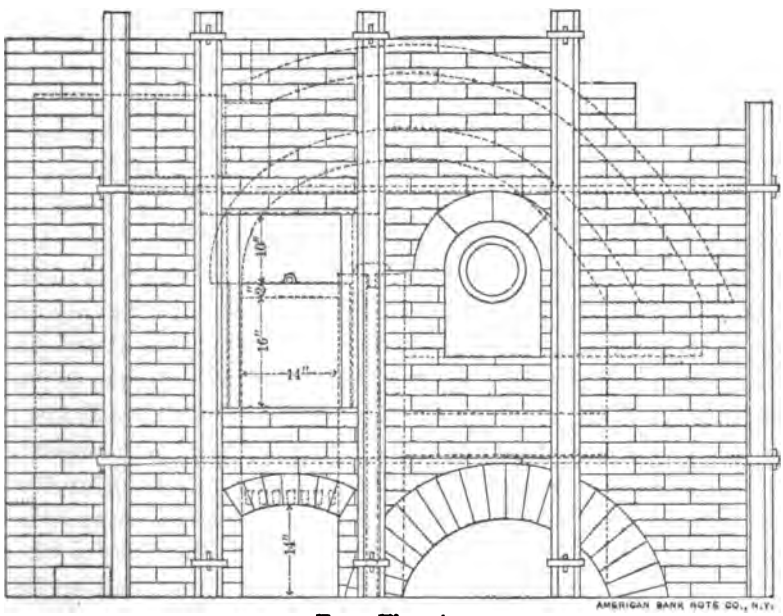


FIG. 5.

Plan with Roof-Arches Removed, Showing Tying-Rods and Projections of Retort, Grate, Fire-Door, Horizontal Flue, and Openings to Arch-Flue.

FIG. 6.



Front Elevation.

can be conveniently managed in laboratory-instruction. The combination of the deep fire-box with the hollow fire-bridge permits a very delicate adjustment of oxidizing or reducing conditions, as well as regulation of temperature. It is accordingly adapted for a wide range of metallurgical operations. Its cost, complete, is about $375, with hearth 36 by 29 inches, fire-box 14 by 36 inches, taking a 9½-inch retort.

Biographical Notice of George W. Goetz.

BY NELSON P. HULST, MILWAUKEE, WIS.

(Chicago Meeting, February, 1897.)

To those who have had the happy privilege of friendship with George W. Goetz, the announcement of his death has brought great sadness.

He was born in Milwaukee, Wisconsin, February 17, 1855, and died January 15, 1897. In his boyhood days, when I first came to know him, his frank, trustful manner; his eyes brimming with truthfulness; his eagerness, never obtrusive, to gain knowledge from all whom he met; and his equal readiness to give without ostentation of his store, constituted a personality which at once commanded confidence and won friendship.

I am told that as a scholar of five years he was conspicuous for brightness, a show-pupil, who was called on to recite when visitors were present at school. Yet this in no way spoiled him. His modesty was too strong a trait in his character to be overcome by any attention which might be expected to arouse vanity. In his simple-heartedness he was pleased then, as he always was in his maturer years, to give of his knowledge for the gratification of his listeners, with never a thought of himself. So strong was this characteristic that in a brief autobiography, in the form of a letter to his wife and children, found among his papers, he excuses himself to them for recounting what he had accomplished in his life-work. As a preface to this autobiography (which, it seems, was prepared on the eve of a sudden journey to California, taken February, 1895, for his health) he says: "I am writing this rather hastily, and I

hope that you will not consider my remarks as vain, for my tendency has always been to rather favor modesty and strive for the ideals of life."

At the age of ten he went to the Engleman private school, where he frequently attended lectures on science, accompanied by experiments. These, as he says in his autobiography, interested him extremely. About this time his mother gave him a small room for his collections of stamps, coins, mineral curiosities, chemical solutions, and an electro-magnetic machine. His interest had been awakened by the scientific lectures at school, and thereafter he began to experiment by himself.

He continued at the Engleman school for about three years, but on account of its expense he had to return to the public schools. He remained in the high school but a year, and from thence went to the University of Wisconsin. Again the expenses of his schooling compelled a change, which took him, in 1870, into office-work as telegraph operator for the Milwaukee Iron Co. in its city office. Here he had much leisure time, which he devoted assiduously to studies, especially of scientific subjects.

It was at this period of his life that my acquaintance with him began. I had just taken charge of the chemical laboratory at the Milwaukee Iron Co.'s mills. To this he was a frequent and most welcome visitor. He was then studying with much diligence and the greatest zest Draper's *Chemistry*, Ganot's *Physics* and Tyndall's books. The subject of electricity possessed intense interest for him. He wrote several articles for the local papers on duplex telegraphy, gave a lecture also on electricity in one of the city churches, and built a telegraph-line, nearly 2 miles in length, to the house of friends, and a private line for one of the tanneries. The latter line, crossing a river, required an automatic contact-breaker and closer at the swing-bridge. For the Milwaukee water-works he devised an electrical instrument which would indicate at the pumping-station the height of the water in the distant reservoir.

But his chief interest was centered in the different departments of the rolling-mills. The blast-furnaces, the puddling-mills, the merchant-bar mills, with Siemens' regenerative furnaces, were all-absorbing delights to him.

It was this environment which enkindled in him a longing

for a thorough technical education along the lines of iron- and steel-manufacture.

In 1876, at the time when Dr. Hermann Wedding came to visit the iron-works of Milwaukee, Mr. Goetz was still in the employ of the company. It having fallen to my lot to show Dr. Wedding about the plant, my young friend asked that he might be introduced to the Doctor in order to make inquiries about the school of mines in Berlin, which he desired so much to enter. I shall ever remember with vivid pleasure the picture of the ingenuous, eager boy and the gracious veteran whose heart was so quickly touched with sympathetic interest by the boy's earnest questioning.

In the fall of 1876 he started joyously for Berlin. His own story tells best of his preparation for Berlin University, and of his student-life at this period. I quote from his autobiographical sketch:

"I had taken mathematical lessons evenings; and to pay for these lessons I gave lessons in telegraphy and short-hand. With $750 I started for Germany. I arrived in Berlin the latter part of October, 1876, rented a small room, and began my studies. I was all alone, and the change from an office-life to quiet student-life was so great. In view of the fact that the most of my knowledge was from self-instruction, and consequently in many directions lacking and incomplete, I could not follow the lectures. This worried me very much. I took an instructor, and by Easter next I was able to follow the lectures. I was so glad, that I celebrated by going to a Wagner Opera!

"I worked late nearly every evening, and when summer vacation came Dr. Wedding kindly invited me to visit him at his summer residence in the Hartz mountains. The fresh mountain-air, the beautiful forests and roadways, made a two weeks' sojourn there very pleasant and beneficial to my health."

His summer vacations were for the most part respites from the class-room or laboratory only. They were spent in visits to numerous metallurgical establishments. In these visits he made the most painstaking study of the various processes of manufacture at the different works, for only a thorough knowledge of anything he investigated ever satisfied him.

Of his last year at Berlin he has written:

"I worked very hard during this year. My dissertation on the 'Production and Consumption of Heat in the Blast-Furnace' was marked 'Very Good.' I was examined in thirteen or fourteen branches, and got as predicate 'With Honor' on eight. Got congratulations from professors. The strain was so great, though, that I was ill for two weeks after that."

Early in 1881 Mr. Goetz was engaged by the Otis Steel Co.

as chemist, introducing at its plant technical gas-analysis to control the producer-gas for the furnaces. The following year he was promoted to the charge of the open-hearth department night-shift. In 1882, in company with Mr. Otis, he made a trip abroad, visiting England, France, Austria and Germany. At Creusot, France, a study was made of mechanical puddlers, and on the return of the party to Cleveland, Mr. S. T. Wellman, the superintendent, gave Mr. Goetz charge of the mechanical-puddling department which, as the result of this visit, was established at the Otis Steel Co.'s works.

His pioneer work in the basic-steel manufacture in the United States is worthy to be recorded in our *Transactions*, and I therefore quote again from his autobiography:

"I had great confidence in the basic process, and was anxious to go to Europe and study it. During my student-days in Europe I had made the acquaintance of Thomas, the inventor of the basic process. He seemed to take much interest in me, and I could have had a position with him in England. Before I left for the United States I spent nearly two months in Hoerde, Westphalia, to study the basic process. At that time Alexander Holley and Thomas were there.

"Mr. S. T. Wellman seeing also the importance of the new process, he and I started for Europe in 1885. Studied the process at Hennabaut, France; also made a general trip to the works in Europe, including Sweden. On our return the process was tried, and I had charge of the experiments. The experiments turned out successfully, and I can therefore claim the successful introduction of the basic open-hearth process in the United States—the first open-hearth heat. It was kept secret by the company that they had made basic steel. Over one hundred and sixty heats were made. I can claim to be the first one who tried and made the basic process a success in the United States. I spent very much time on the subject; wrote many reports on it."

In 1886 he went back to Berlin to marry Miss Elsie Luedecke, whom he had met at the home of his devoted friend, Dr. Hermann Wedding.

He continued his connection with the Otis Steel Co. until 1888, when he received a commission from Mr. George Westinghouse to visit Europe for the purpose of making a report on gas and fuel and all improvements and economies relating thereto, to be followed up on his return by experiments in Pittsburgh on the possibility of producing a cheap and efficient fuel-gas for distribution.

This investigation took him to Sweden, England, France, Austria and Germany. He had charge of all the experiments made at Pittsburgh, which were conducted to the best advan-

tage, by reason of his extensive investigations abroad. Of his work at that time he says:

"The anxiety and difficulty necessarily connected with experiments on so large a scale were very telling on my health. Had to write and study evenings a good deal. As soon as I had demonstrated what could and what could not be done, we stopped all experiments."

In Vol. xviii. of the *Transactions* he has given in "Notes on Fuel Gas" some of the many facts which his investigations for Mr. Westinghouse brought out.

In 1890 he removed to Milwaukee, having made a business connection with Mr. Ferdinand Schlesinger, who had become a large mine-operator in the Lake Superior region. The new duties proving distasteful to him, he resigned his office the following year. Mr. Goetz visited Europe during this year with commissions to make certain scientific investigations.

On his return to America he was appointed consulting metallurgist to the Illinois Steel Co., the Westinghouse Co., and the Wellman Iron and Steel Co. Having become established as a consulting engineer, he provided himself with a very complete chemical and electrical laboratory, in which he could undertake experimental work involving the treatment of large samples, and the most elaborate investigations. In 1894 he was again commissioned by Mr. Westinghouse to visit Europe, this time in company with Mr. S. T. Wellman, for the purpose of scientific investigations. This trip of three months was for the most part a weary one for Mr. Goetz. With all his enthusiasm for studying new processes, and with all his keen delight in the acquisition of knowledge, he was continually oppressed and disheartened by a languor which his strong will could not resist. Immediately on his return home he consulted a Chicago specialist, and learned that his malady was mortal. With brutal frankness this physician told him that at the best he could live but a few months. The shock completely prostrated him, swept him like a wrecked mariner into the depths of despair. Through it all, the sublime, unfaltering courage of his wife cheered and sustained him. The infection of her bravery won him back to hope, and with it came gradually returning strength in slight degree. A rigid diet and change of surroundings, by a visit to California, further benefited him. He returned to Milwaukee in the summer of 1895, resuming then,

in a moderate way, some of his duties to his several clients. He also busied himself, so far as strength and time permitted, in elaborating a chemical method, devised by him in 1894, for extracting copper from the tailings of stamp-mills, as well as from ores. That he might escape the inconvenience of a slight physical disability, he submitted on Monday, January 11th, to a simple surgical operation. Until Thursday following he seemed to be rapidly recovering from it, but on that day his old malady in its most acute form set in, and on Friday noon he passed quietly away.

In this record of his busy life one sees how tireless he was in his studies, for he was ever thus engaged. He was just as unsparing of himself in his service to others. It was always his very best that he gave. Being scrupulously conscientious, and succeeding so well in the tasks given him, he invariably won the confidence of his employers and of his associates. Although he was overburdened with regular duties in his business engagements, he yet found time to devise some eighteen patents, nearly all of which were in the domain of metallurgy. The friends of Mr. Goetz, familiar with the work he had accomplished, can but mourn that he was stricken down so early, before he had reached the prime of life. With an experience so full, with a mind so active, so well-trained in habits of investigation, there was ample reason for hope that he would continue, as he had begun, to do notable work in his chosen lines. In 1892 he was elected one of the managers of our Institute, and he also served actively and efficiently as one of its committeemen in connection with the memorable International Chicago meeting at the time of the Columbian Exposition. His extensive acquaintance among foreign metallurgists made his services on this occasion especially valuable. Strangers were quickly drawn to him, for the childlike simplicity of the man disarmed whatever of reserve they might have. It was easy to perceive that his ideals were lofty, so transparent was his nature, and his ideals were not indulged in as dreams, but wrought out in his daily life—a life of great sweetness and purity.

Postscript.

S. T. Wellman, Cleveland, O.: I can add but little to what Dr. Hulst has said about our dear friend, George W. Goetz.

His history of Mr. Goetz, up to the time of his coming to Cleveland, agrees in many details with what he himself told me during our long acquaintance, which commenced in 1881, when he came to visit the Otis Steel Works with Mr. J. J. Hagerman. As soon as I met him, I was at once attracted and impressed by his earnest enthusiasm and his evident thorough knowledge of chemistry and metallurgy. Although we had no vacancy in the works, I engaged him at once, and made a place for him. He soon became very valuable to the company. He started in at the laboratory. His work was thorough and conscientious, and could be relied upon at all times. He was always enthusiastic, especially when any metallurgical problem came up. From the laboratory he was given charge of one shift in the open-hearth department, part of the time being on night duty. The work which he accomplished in this position was very satisfactory to me, and the steel turned out was particularly uniform. As mentioned by Dr. Hulst, Mr. Goetz visited Europe with Mr. Otis in 1882, and on their return the building of a mechanical-puddling department was started. Two furnaces, each capable of puddling 3000 pounds in one charge, were built, and the plant was given in charge of Mr. Goetz. His work in this department was characterized by his usual enthusiasm and energy. No one who has not been through such an experience can form any idea of what the starting of such a plant means to the one in charge. New men had to be broken in, and new problems had to be faced every day. In this plant the iron to be puddled was melted in a 15-ton open-hearth furnace. The furnace was charged until it was filled. The melted iron was then raised to steel-melting heat. Then the problem was to draw off safely 3000 pounds at a time into small ladles. It could not be tapped out in the ordinary way, because it is impossible to stop off a stream of iron safely which has behind it a head of melted iron, in some cases of nearly 2 feet. After many mishaps and hard work, the problem was solved by Mr. Goetz by a series of tapping-hole bricks placed close together, one above the other. The first iron was drawn out of the upper hole and so on, each hole being opened in succession, until the furnace was emptied. The plan worked very successfully, and was patented by Mr. Goetz. He had charge of the mechanical puddlers as long as they were

operated. This was until the advent of the Bell-Krupp process, which made it possible for the Otis Company to buy washed metal as cheap, or cheaper, than they could make puddled blooms.

Dr. Hulst has spoken of Mr. Goetz's pioneer work in making open-hearth basic steel. He always had the greatest faith in the future of the process; and when the Otis Company, after the visit of Mr. Goetz and myself to Europe in 1885, concluded to allow us to try the experiment, he went at the work with his usual enthusiasm. The experimental furnace was started in January, 1886, and run until April of the same year. Much to Mr. Goetz's regret and disappointment, the experiments were then discontinued, the company's reason for which was that the demand for their product was so heavy that they could not spare the furnace for further experiments. This was undoubtedly a great mistake, and was so viewed by Mr. Goetz and myself. He severed his connection with the Otis Steel Co. in 1888. I advised him to take the step, as I believed it would open up a wider and more remunerative field for him. The wisdom of the change was soon seen, as it was only a short time before his income was doubled. His work, and the reputation which he obtained after establishing his laboratory in Milwaukee in 1890, soon became world-wide, and he was believed by all of his friends to be one of the best, if not the best, all-round metallurgist in the United States.

I knew him probably as well as any one, outside of his own family, and I know he would sooner have cut off his hand than sign his name to any paper that was not true in every detail.

Never shall I forget the last trip we took together. It was made to Europe, in the interest of the Philadelphia Company of Pittsburgh, Pa. Throughout all this journey, as Dr. Hulst says, he was continually oppressed by a languor which he could not resist, and which neither of us could understand. If we had only known then what the trouble was, I doubt not that he would be alive to-day. But it was not to be. He was a noble fellow, honest as pure gold, and his life was a good example of what can be done by a naturally bright, energetic boy, who, although poor, is bound to have knowledge, and who, in spite of all obstacles, succeeds. Mr. Goetz was naturally a student and a born investigator. He possessed a wonderful

memory, and was a good talker on any subject in which he was interested. This made him most delightful company. He leaves a noble young wife, whose courage and cheerfulness had much to do with prolonging his life for the last two years. His three bright, beautiful children, two boys and one little girl, promise to grow to be all that their father could have wished. He was a loyal friend to me through many hard trials and troubles, and I never expect to see his like again.

Biographical Notice of Peter Ritter von Tunner.

BY R. W. RAYMOND, NEW YORK CITY.

(Lake Superior Meeting, July, 1897.*)

THE death of the eldest of the honorary members of the Institute was not unexpected. Ritter von Tunner had never recovered from the effects of a paralytic stroke received in 1892, and when the same enemy struck again in February of this year the blow was plainly mortal. He passed away June 9th, gently and without pain. Thus ends a career of great distinction and fruitfulness, crowned with fame and love. This pioneer of the age of iron and steel had lived so long and wrought so much that the present generation could scarcely recall the whole category of his services. There can be no more appropriate use of the pages of these *Transactions* than to honor them with a record of such a life and work, for the inspiration of other lives to other labors.

Peter Tunner was born May 10, 1809, near Peggau, in Styria.† In the same province, his grandfather, Josef Tunner, had purchased a primitive nail-forge and enlarged it to a ham

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* Ritter von Tunner died June 9, 1897, and the news of his death reached this country too late to permit the preparation of this notice for actual presentation at the Lake Superior meeting of the Institute. But it is appropriately included among the papers of that meeting.

† For the material of this notice, I am indebted to a sympathetic and comprehensive obituary article by E. Heyrowsky, which appeared in an extra supplement to No. 7 of the *Vereinsmittheilungen* of the *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, of this year.

mer-works. His father, Peter Tunner, was in 1809 owner of a blast-furnace as well as of the inherited hammer-works. The boy became, consequently, familiar with the manufacture of both pig- and wrought-iron, even before he had completed his primary education in school. In his twelfth year he entered the high school at Graz, from which he was graduated with honors two years later. Meanwhile his father had been obliged, by reason of the lack of ores, to abandon his own blast-furnace, and had taken, in 1823, the position of mine- and furnace-manager for Prince Schwarzenberg, at Turrach, Styria, where the son joined him after graduation, and worked under him, partly at the blast-furnace in Turrach, partly at other iron- and steel-works of Prince Schwarzenberg, acquiring a thorough and manual acquaintance with all the processes then and there in use. So completely did he master the theory and technique of the business that, four years later, in the eighteenth year of his age, he was engaged by the Brothers von Rosthorn, a firm of iron-manufacturers in Carinthia, to reconstruct and modernize their whole system of metallurgical operations. Well satisfied with the results of this experiment, they rewarded the young ironmaster by providing him with free board and lodging in Vienna, and thus enabling him to gratify his cherished desire for a course of study at the Polytechnic Institute in that city. Here, as everywhere and always, his ability and zeal won hearty recognition. Besides taking with honor the regular course at the Institute, he studied, outside of it, mineralogy and geognosy under Riepl and mineralogy under Mohs. The position of an assistant in the Polytechnic Institute was offered to him, but declined. So was an official position in the puddling-works at Witkowitz—the first of the kind in Austria. He preferred, in accordance with his father's wish, to return to Styria, and to his former work at Turrach. But his activity in this position was interrupted by a serious illness, after which he employed the period of a slow convalescence in arranging an extensive mineral cabinet belonging to one of his former generous patrons, Franz von Rosthorn.

At the end of 1831, being not yet twenty-three years old, he took charge of a neglected iron-works at Mauterndorf, in the Salzburg district, which he very rapidly restored to an effective activity, and in 1832 he became manager of the newly-built

hammer-works of Prince Schwarzenberg at Katsch, near Murau, Styria. The three years which he passed here he was accustomed to recall as among the happiest of his life. The young master was adored by his workmen, and, aided by their loyal service, soon made the establishment famous, so that crowds of young men (including graduates from Schemnitz, then the only Austrian mining academy) trooped to Katsch for the benefit of practice under such a guide. Thither came also, in 1834, the Archduke John (beloved patron of Styria), begging Tunner to accept a professorship of mining and metallurgy which was to be created in the Polytechnic Institute of Graz. This offer, extended to a youth of twenty-four, reflected credit upon the merit of him who received, and the discriminating insight of him who made it. The Archduke was not even discouraged by Tunner's declination, upon the ground that he had never looked forward to a professorship, and had not studied sufficiently all the branches which it would represent. Very likely this modest hesitation betrayed a regretful ambition. At all events, it was finally arranged that Tunner should receive the professorship and with it a furlough of two years, and an allowance of 10,000 florins, for travel and study in the principal mining and metallurgical centers of Europe. In March, 1835, the decree appointing him was issued. It was determined, however, that the new mining school should be established, not at Graz, but at Vordernberg, then the center of the Styrian iron industry.

Prof. Tunner's travels during the next two or three years were extended through all parts of Austria and into North and South Germany, Sweden, England, Scotland and France. He spent some months at Freiberg, in Saxony, where he took lessons from Plattner in blowpipe-assaying, and became intimately acquainted with the group of great teachers then gathered at the Freiberg Bergakademie, such as Weisbach, Reich, Kersten and Gaetzschmann. At Berlin he worked in analytic chemistry under Heinrich Rose and Mitscherlich, and in mineralogy and geognosy under Gustav Rose and Dechen. In London he entered the laboratory and heard the lectures of Faraday; in Falun he was a pupil of Sefström. Everywhere he took notes and collected specimens, especially of ores and furnace-products.

Upon his return from this laborious and useful tour, he found

Peter Tunner

that the buildings for the new school were far from ready; and for a year or more he made Vienna his headquarters, busying himself with the arrangement and enlargement of his own gathered experiences, and especially with the earnest endeavor to introduce into Austrian iron and steel practice the improvements which he had observed in other countries. For this purpose he visited many iron-works and published several important papers. The first, a brochure on "The Use of Heated Blast in Iron Metallurgy," was followed by larger treatises on "The Present State of the Puddling Process and its Relation to the Hearth-Refining Process of Inner Austria," "Contribution to the More Accurate Knowledge of the Chemical Composition of Refinery Slags," "Rolls as Substitutes for Hammers;" and "Contributions to the Investigation of Possible and Suitable Improvements in the Austrian Hearth-Refining Process." In 1839 he spent some time at the Schemnitz mining academy and among the mines of Hungary, and visited also the iron-works of Austrian Italy.

At last, the preparations at Vordernberg being complete, Tunner began his work there, opening the new school November 4, 1840. The entire faculty of instruction consisted of this one man, then 31 years old, but already well-known for comprehensive knowledge and practical skill. For eight years he gave instruction, one year in mining and the next year in metallurgy, so that each course was given four times. Besides the general scientific exposition of principles (with special reference to coal-mining and iron metallurgy) the practical training of the students was provided for. The daily programme was one and a half to two hours of lecture in the forenoon, followed by the completion of notes and drawings by the student, and in the afternoon, a colloquial discussion of the same subject, or (once or twice a week) excursions to neighboring mines and works, of which the students were expected to make written reports. Every Saturday the teacher read and corrected these reports. After the course of lectures, the mining students spent two or three weeks in mine-surveying and mapping, and four or five weeks in manual practice of the mines. The metallurgical students had a corresponding course of five or six weeks in an iron-refinery built for this purpose by the government (*Lehrfrischhütte*), or at neighboring iron-works. All this

was done under the direct personal supervision of Prof. Tunner, assisted only by unpaid volunteers among the mining captains and furnace-foremen.

The lectures and practical courses occupied the school-year from November 1st to the end of June. Then began the grand tour of about five weeks, among more distant mines or works. This was a continuous study under the eye of the teacher. Written reports were made of it by the students, corrected by him, and included among the data determining the final standing of each upon graduation. Thus a single tireless and enthusiastic pioneer devised and executed a plan of combined theoretical and practical training which the great modern schools have done little more than imitate. The graduates of Vordernberg were eagerly sought by mine- and furnace-owners; and their rapid promotion and successful performance of professional duties increased in turn the reputation of the school and its head.

As already observed, Prof. Tunner carried on, alone, this amazing and admirable work for eight years, that is, until the end of the school-year 1847–8. In 1848 the revolutionary condition of Hungary necessitated the closing of the mining academy at Schemnitz. The greater part of the students migrated to Vordernberg, and the government transferred a professor of mining also, so that the two courses could be given simultaneously. At the same time, the school, which had been only a provincial one up to that date, became a "provisional Royal Imperial" one. But the accommodation of the numerous additional students and the extension of educational work proved a difficult matter in the little mountain village; and the school was moved to better quarters in Leoben, where it opened November 1, 1849, as a permanent (no longer "provisional") Royal Imperial Mining School, having a professor at the head of each of the two courses, assistant instructors, and Tunner as director of the whole. It has remained at Leoben ever since. In 1861 it acquired the well-deserved rank of a "Mining Academy." Throughout its splendid history, this institution has borne the impress and has been inspired by the enthusiasm of its great founder. For many years, though relieved from the exhausting detailed duties of instruction which he had discharged at first, he continued to deliver the

lectures and conduct the practice on the metallurgy of iron and steel, attracting a crowd of students from all countries. In 1866, under the pressure of the military disasters and consequent financial difficulties of that year, the Austrian government reduced the courses at Leoben, temporarily crippling the institution. In 1871 the lost ground was recovered; but in the meantime, Tunner had withdrawn from the lectureship on iron and steel, and had become the representative of the town of Leoben, first in the Provincial and afterwards in the Imperial Parliament. In this capacity he had opportunity to do valiant service for his beloved school, which was in danger of being transferred to Vienna. His earnest protests and arguments were successful; and in 1874 the question was definitely settled, and the academy was left at Leoben. Somewhat earlier in the same year Tunner was formally retired upon pension, at his own request—an event which was celebrated in November, 1874, by the so-called *Tunner-Feier*, a gathering of representatives of the Imperial and Provincial governments, technical faculties and industrial corporations, but chiefly of the former pupils of the distinguished master, who had for thirty-four years so brilliantly administered the institution of his own creation. A portrait-medallion, struck in honor of the occasion, was presented to him, together with a loving-cup. It was in accepting these tokens that the venerable Nestor of mining and metallurgy said: "When, some day, my last hour shall come, I know not whether my thoughts will be more of my family or of my academy and my pupils."

But this ceremonial retirement was the beginning of more than twenty active and fruitful years. Indeed, Tunner's release from the duties of administration left him free for the wider usefulness for which his ripe experience had qualified him. He followed with tireless interest all new developments, particularly in the metallurgy of iron and steel, and promoted mightily the adoption of useful improvements by his countrymen. While director at Leoben, he had taken a foremost part in the introduction of the hot blast, and the manufacture of cement-steel, cast-steel, and springs. But he chiefly distinguished himself by his intelligent and ardent advocacy of the Bessemer process, even in the day when its practical value seemed problematical to many high authorities. The early

Bessemer plants at Turrach and Heft owed their origin to Tunner's urgent recommendation and were inaugurated under his personal supervision. Those of Graz, Neuberg, Ternitz and Zeltweg, which rapidly followed, made Austria a home of the new art, and pilgrim-students from other countries sought the Austrian Alps to study the practice there exemplified.

It is in fact difficult to say in which of the two spheres of his activity—that of the thorough and inspiring instructor, or that of the traveler, observer, critic and professional adviser—Tunner's work was more remarkable or more productive of lasting benefits to his country and his profession. Certainly he could not have given forth so much, either of knowledge or of influence, if he had not continually refilled his own mind by assiduous study and observation. It goes without saying that he was a regular visitor at international expositions, and that his visits bore fruit in valuable reports. Usually he was either a juror or a commissioner from his government. The exposition at Munich (1845), London (1851), Paris (1855, 1867 and 1878), St. Petersburg (1870), Philadelphia (1876) and Düsseldorf (1880), all received the benefit of his co-operation and presence, and enriched in turn his contributions to technical literature. In 1857 he made a second journey in Sweden, and published, a year later, a book on Swedish iron-metallurgy. His work on the iron and steel manufacture of the United States (1877) is classic, though it has been, like other classics, left behind by the rapid advance of the art it depicts. His book on Russia's Mining Industry was the result of his visit, at the invitation of the Russian Government, to the St. Petersburg exposition.

In the introduction (1880) of the Martin open-hearth process into Austria, the veteran took an active part, personally directing, in that year, the works at Donawitz near Leoben where his son Ludwig, with his aid and counsel, subsequently conducted the basic open-hearth.

After the death of his wife, in 1881, his health appeared to be more or less broken. In 1887 he was brought near to death by an inflammation of the lungs, from which he slowly recovered. In 1890 he was able to take part with vigor and enthusiasm in the semi-centennial celebration of the Leoben Academy, which was attended by 300 of his former pupils, and naturally

enough became practically an ovation to the venerable master. By a happy coincidence, the international sessions connected with the Fifty-Seventh Meeting of the American Institute of Mining Engineers (of which Tunner had been elected in 1873 an honorary member) were in progress at the same time in Pittsburgh, Pa., and by the unanimous vote of the representatives of this Institute, the Iron and Steel Institute, the *Verein Deutscher Eisenhüttenleute*, and other technical societies there assembled, a telegram was sent to Leoben, conveying cordial greetings and congratulations to the institution and to its founder.*

For more than a year longer, Tunner continued his literary activity. As late as February, 1892, he wrote me a letter, a translation of which I incorporated in the *Transactions*,† concerning the use of carbon-bricks in lining blast-furnace hearths. This communication illustrated the characteristic quality of his whole career. To the very last the veteran marched in the van of practical progress.

In December, 1892, he experienced a stroke of partial paralysis, from which he never completely recovered, though he attended meetings of the mining institute of Styria and Carinthia in Leoben until 1895, and was still corresponding with friends and colleagues in 1896. In February, 1897, came a second stroke, which proved fatal; and, lingering until June, he passed painlessly away.

His brilliant and useful life did not fail of contemporary recognition. He was loaded with honors well deserved and modestly borne. Besides the regular successive promotions which he received in the service of the Empire (Sectionsrath, Hofrath, etc.) and the hereditary rank of Ritter, conferred in 1864, he was decorated with numerous orders by the sovereigns of Austria, Sweden, Würtemberg, Saxony, Bavaria, Prussia and Russia, and with marks of distinction not less valuable in his eyes, such as the Bessemer medal, the honorary membership of the Iron and Steel Institute, the American Institute of Mining Engineers, the American Philosophical

* See *Trans.*, xix., xxii. The message was as follows: "The International Session of German, British, and American Engineers and Metallurgists send greetings to Tunner and Leoben."

† *Trans.*, xxi., 120. See also *Trans.*, xxvi., 185.

Society, the honorary citizenship of Leoben and many other Styrian and Carinthian towns, and membership in many scientific and technical societies. But the best reward he could have asked for his long, faithful and fruitful service is the one upon which he can still look with satisfaction, now that all echoes of earthly praise have died away—the picture of a great national industry reconstructed and rejuvenated through his wise and tireless endeavors, and of a host of earnest laborers, guided by his teachings and inspired by his example.

Improvements in Mining and Metallurgical Appliances During the Last Decade.

BY E. GYBBON SPILSBURY, TRENTON, N. J.

(Presidential Address at the Chicago Meeting, February, 1897.)

In the course of the persistent and rapid advance of our country towards the goal she has set for herself, of commercial and manufacturing supremacy, there stand out certain periods or cycles of prosperity or depression, succeeding one another with a comparative regularity as to their ultimate trend, but of varying lengths and intensities. I might compare this course to that of a railroad between two points at different levels, with intervening mountain ranges and valleys, the up-grades climbing towards the final upper terminal of prosperity, while the down-grades retard the ascent, and tend to return to the valleys below. Just at the present time we are evidently at the bottom of one of these valleys, and, before starting to climb the next divide, it may be prudent for us to look into the condition of our rolling-stock, and especially into that of our brakes, and see what the experience of our downward journey has taught us, and whether we are properly equipped for the new upward advance which now lies before us.

Probably no previous period of the same duration has produced in all branches of mining and metallurgy improvements so many and so various as have the last ten or twelve years. Not that they have brought forth any such startling and revo-

lutionizing inventions as were those of Sir Henry Bessemer, or Thomas and Gilchrist, in the metallurgy of steel. There has been rather a continued series of improvements and economies in lines already laid out. These have resulted in enabling the miner and manufacturer so to increase his output and lessen his cost of production as to be able to meet prices and conditions of trade wholly unexpected and unprecedented.

Iron and Steel.—Let us take the iron and steel trade for an example. Ten years ago it would have been considered an impossibility to produce Bessemer pig to sell at a profit below $17 a ton, and foundry pig at $12 a ton was equally impossible. To-day the market-price for the former is as low as $11, while ordinary irons are selling in the South below $6 a ton at furnace, and the producers are not actually losing money at these figures. I do not mean to say that the whole of this change has been brought about by improvements in mining and smelting. We must recognize that at least a portion of the saving is due to the discovery of enlarged sources of cheap ore-supply. At the same time, the improved methods employed in mining these large deposits have enabled the owners to place their ores on the market at figures which, a few years before, would have been considered impracticable. The introduction of the steam-shovel for the mining of the enormous hematite deposits of the Southern States, and more recently of those of the Mesabi range in Minnesota; the application of electrical power for lighting, mining, concentration and transportation, and the many improvements introduced in the various systems of wire-rope haulage and transportation, have all been the means of bringing down the cost of producing iron-ore to figures undreamt of ten years ago.

The change in furnace-practice has been no less radical and progressive. In localities where, twelve years ago, a production of 1000 tons per week was considered far above the average, it is now deemed to be below a profitable producing basis, and the average may be taken as 2500 tons per week, while some furnaces (those at Duquesne) are now reported as producing as much as 6500 tons in the same time; and these furnaces are now aiming at a regular output of 1000 tons per day. As already observed, this increased production has been made possible, not by the invention of any one individual, but rather

by the industry and intelligence of the vast army of toilers in every branch connected with the industry. The call for increased heat in the air-blast was met by improvements in the design and construction of the fire-brick stoves. The higher pressure made necessary by the increased height of the modern stack, together with the growing use of larger percentages of fine hematite ores and concentrates in the charge, soon called forth the necessary improvements in the design and power of the blowing-engines; while the danger of a too rapid cutting-out of the furnace-lining, due to the increased heat around the zone of fusion, has been provided against by improvements in the water-jackets, which, from being merely a protection around the tuyeres, have developed into a protection of the whole furnace below the bosh-line. At the Duquesne furnaces, again, they are now doubling the number of tuyeres (making sixteen in all), it having been found impossible to supply the requisite amount of air through one set. Moreover, the rapid driving of the modern furnace has necessitated the abandonment of the old methods of charging by hand-barrows, and the substitution of automatic hoisting- and charging-devices capable of handling from 1700 to 3000 tons of material per day.

While it would be impossible in a paper as general as this must necessarily be to specify the many inventions which have tended to all these advances in the metallurgy of iron, there are one or two which deserve special mention, since they enable the further use of the iron produced with results heretofore unattainable. The first of these is the mixer, invented and patented by our late member, Capt. William R. Jones, of Pittsburgh, by means of which the products of one or more furnaces are tapped into a receiver, and there so thoroughly agitated and mixed as to furnish an absolutely uniform metal for further treatment in the manufacture of steel. By allowing the fluid metal to remain a certain time in these mixers (as is the custom in this country), or by the further use of lime (as is the custom in some of the English works), the resulting metal is very thoroughly desulphurized, and the time necessary for its conversion in succeeding processes is materially lessened.

Another improvement is the casting of the pig-metal in chills instead of into sand direct, thus materially lessening the

silicon in the remelted metal, and furnishing a material specially adapted to the basic open-hearth process.

A third improvement, while it has not yet come into such very general use, is, I imagine, destined to play a very prominent part in the steel industry of our Southern States. I allude to the desiliconizing process invented by Mr. Benjamin Talbot. This process consists in pouring the metal as it is tapped from the furnace through a bath of molten oxide of iron, whereby almost all the silicon contained in the iron is eliminated.

If the advances and improvements have been so great in the production of the pig-iron, they have also been equally so, if not greater, in the conversion of the pig-metal into the different grades of steel. This is, perhaps, not so apparent in the Bessemer process, which had already attained almost its present perfection at the beginning of the period covered by this paper. It is true that closer attention to the mechanical details of the handling of large masses quickly, and to the requirement of using as uniform a charge as possible, has greatly lessened the time necessary for conversion, and has thus enabled the works which have availed themselves of this knowledge to increase their product greatly, while lessening its cost; but it is in the open-hearth processes, both acid and basic, and especially in the latter, that the longest strides have been made. The economies effected during the last few years in open-hearth practice have been so great, that to-day steel can be produced by this method practically as cheap as by the Bessemer process, and of far better and more uniform quality. While, ten years ago, the difference in the cost of production by the Bessemer and the open-hearth was between three and four dollars, the real difference to-day is not much more than fifty cents a ton; and it is not presumptuous to predict that, in the near future, even this small excess will be wiped out altogether. These economies, most of which have been in the line of mechanical rather than metallurgical improvements, have made it possible to increase the size and the output of open-hearth furnaces to an extent undreamt of five years ago. The mechanical charging-machines invented by Mr. S. T. Wellman make it as easy now to operate a twenty-ton furnace as it was ten years ago to run a five-ton one; and while twenty-ton furnaces are still about the average, there seems now to be no limit to what may be

done by the introduction of the Wellman revolving furnaces, which are already operated with fifty-ton charges at the Illinois Steel Company's Works in South Chicago, where the erection of a new seventy-five-ton furnace is even now contemplated.

The development of the basic process, both Bessemer and open-hearth, has only quite lately become general in this country, notwithstanding its very wide introduction in Europe more than ten years ago. The reasons for this slower development are manifold. In the first place, the mechanical genius early developed by many American engineers at the head of the Bessemer works in this country, from among whom it certainly will not seem invidious to single out the name of our late member, Alexander L. Holley, rapidly brought the Bessemer process to such a state of metallurgical perfection and economy of production as to make it seem unwise to those having capital to invest, to introduce a rival process in which the cost of production was at that time necessarily much higher. A second reason was probably the natural reluctance of the enormous capital already invested in Bessemer plants, under the protection of the Bessemer patents, to branch out into other developments, which were, on the one hand, not so well protected by ground-patents as was the Bessemer process, and the resulting product of which, on the other hand, would, by its more uniform character, rapidly replace the Bessemer product. A third very apparent reason appears to me to have been the fact that in this country it has not been the exception, but rather the rule, to find our largest iron-ore deposits adaptable to the Bessemer process, whereas England, Belgium and Germany have had to depend to a very great extent on the importation of foreign ores, sufficiently high in iron and low in phosphorus to act as a corrective admixture for their own high-phosphorus ores, if any of the latter were to be used at all. At the present time, however, all these reasons have given way to the influence of the improvements mentioned above, and present developments all seem to tend in the direction of the basic open-hearth. The improvements in basic hearths and linings made by Mr. Benjamin Talbot have done away, by the use of crushed crude limestone and resin, with half the expense formerly required for this purpose.

At this moment our English neighbors are considerably

excited over the possibilities of the Bertrand-Thiel continuous open-hearth process, which has found a most enthusiastic advocate in Mr. Gilchrist. Whether, with all the advantages claimed for it, it will be able to compete either in output or in economy with the Campbell or the Wellman revolving furnaces, remains to be seen.

Another recent development in England which seems to have some promise for small plants making special grades of steel, is that known as the Stockman process, which, by the use of nitrate of soda as a means of oxidizing the objectionable metalloids, supersedes the necessity of expensive blowing-machinery. The process is highly spoken of by good authorities who have seen it in operation; but so far I have heard of no converters larger than one-ton capacity being used, and the resulting metal appears always to have been recarburized in an open-hearth furnace. What the final cost of these double processes will turn out to be has yet to be determined.

I cannot well leave this branch of my subject without calling to your attention the remarkable work in the mining, crushing, concentrating and finally briquetting of magnetic iron-ore which is carried on at the Ogden mine in New Jersey by Mr. Thomas Edison. As a theoretical development and improvement of the very highest type, these works of his stand out unique; but it has not yet been demonstrated in regular practice that the enormous outlay called for by such a complete plant (the present plant at Edison being reported as involving already an expenditure of one and one-half million dollars) would be warranted except in very exceptional cases. The experiments so far carried out on some 30,000 tons of rock have demonstrated the fact that an ore averaging not over 30 per cent. of iron can be quarried or mined, transported by rail a quarter of a mile to the mill, there crushed, sized and re-crushed to a fine powder, the magnetic iron separated from the non-magnetic gangue, the concentrates mixed with a binder, then pressed into briquettes, furnace-dried and loaded on cars —all for less than six cents per unit of the iron-contents of the briquettes. Even under the present conditions of low prices, if the resulting concentrates were low enough in phosphorus to bring them within the Bessemer limit, the operation could be carried on to a financial success; but unfortunately this is not

the case, and therefore, until the price of foundry-pig shall advance to a figure at which it will be profitable to purchase a 68-per-cent. iron-ore at six cents a unit, it is not probable that the Edison works can be run continuously at a profit. It stands, nevertheless, a monument of perseverance in original research which certainly deserves our admiration.

Copper.—When we come to the mining, smelting and refining of copper, we find that no less progress has been made, and mines which, ten years ago, were practically abandoned as being too poor and expensive to operate at a profit, are to-day yielding good returns to their owners. Instead of operating on a small scale with the rich ore-streaks in a vein, it is now found much more profitable to work the whole vein, notwithstanding the fact that the average yield of copper throughout a deposit so worked may not yield over 3 or 4 per cent., and even (in the case of the Atlantic mine, at Lake Superior) does not reach 1½ per cent. The average cost of producing copper to-day is certainly under 10 cents a pound, and many of the larger mines are able to furnish it at a cost of less than 7 cents. It is hard for anyone not actively engaged in the metallurgy of copper to be able to point out any one special invention or improvement which has contributed more than another to these results, especially as the characters of the ores and their methods of treatment are so thoroughly different in the two great copper-producing districts of the United States. In the great Lake Superior deposits, the copper being entirely in a native state, the problems to be solved for cheaper production have been chiefly mechanical ones. Here, therefore, these improvements have taken the direction of handling enormous quantities of material with the least possible labor. Deeper shafts, increased size and speed of hoisting-engines, and improved facilities for mechanically crushing and concentrating the mine-product have permitted wonderful savings to be effected. Probably nowhere else has the mechanical engineer been able to supplement so thoroughly the work of his mining confrère. Nowhere else have the greatest economies of the steam-engine been more thoroughly studied and applied than in this district. Triple and quadruple expansion hoisting- and pumping-engines are now the rule, instead of the exception, and the enormous steam-stamps, with their almost incredible capacities, are trophies of

the engineering skill of those who have evolved them from the pattern of the original Ball stamp. The improvements in the concentration of the crushed material have kept equal pace with the rest of the development until, to-day, a typical concentrating-mill of this district, handling a thousand tons or more a day, is operated with less labor than an ordinary 100-ton mill elsewhere. In the western copper-regions of Montana and Arizona these problems have also had to be met, and have been similarly overcome to a greater or less extent; but, in addition to them, the complex and often refractory character of the ores met with has raised still other difficulties, both in their concentration and in subsequent metallurgical treatment. The successful matting of these complex ores, their subsequent enrichment by the Bessemerizing process, and, finally, the parting and deposition of the copper by the several electrolytic processes, are all inventions and improvements which have come into general use during the last decade, and by these means this country has been enabled to control practically the copper-markets of the world.

Gold.—If we consider the mining and production of gold, we shall be still more struck with the advances which have been made. In view of the fact that practically all the gold-production of the world is from native gold, it would not seem that, outside of the mechanical methods employed in mining, crushing, amalgamating and concentrating, there was much scope for new inventions or processes; but, in truth, very few branches of metallurgy have been so thoroughly revolutionized during recent years. Where, a dozen years ago, a mine which yielded anything less than one ounce of gold to the ton would hardly have proved attractive to the most sanguine investor, to-day many of the best paying mines are working on ore containing less than four pennyweights of gold to the ton; and even this is not necessarily all in a free state, the most of it being contained in iron sulphides, which require at least a partial, if not complete, roasting before treatment. The necessity of the close and careful concentration of these small quantities of auriferous iron sulphides from the immense volume of quartz or slate gangue in which they occur has been met by the improvements in the different continuous belt-concentrators, which are the natural evolution from the old Rittinger or *Stossheerd* of

twenty years ago, aided by the improved hydraulic sizers or separators. It is in the treatment of these concentrates and the extraction of the gold they contain that the greatest improvements have occurred. While in some cases it is still considered economical, where extremely close desulphurization is required and where fuel is very cheap, to continue the use of the hand-rabbled reverberatory furnaces, such cases are in the minority. The mechanical furnaces have been so much improved and so greatly increased in capacity that they are practically replacing all others. There are so great a number of these that it would take too long to enumerate them. I will therefore only mention one of the latest, and probably the best of them all. I refer to the Pearce turret-furnace, which is such a radical departure from the general furnace-construction as to call for a few words of description. Before this furnace was devised, it had been the general aim of inventors to employ mechanical contrivances adaptable to the construction of the then existing rectangular hearths. These contrivances all had the defect of being cumbersome and difficult to repair without stopping the whole operation. In the Pearce type of furnace the hearth is the rim of a circle and the mechanical stirrers can be driven from a central point, and so protected by water-cooling as to prevent their rapid wear by oxidation. The speed at which the ore is carried round the furnace can be regulated with exactness, and thus constant results are obtainable. The ore-feed is also regulated by a mechanical device, and so is the firing, which can be either by direct fuel or fuel-gas, as desired. The saving in labor on furnaces of this type is also very considerable.

Of the numerous inventions and processes which have been brought before the public in the last ten years for the treatment of these roasted (or even unroasted) concentrates, only two have stood the test of practice and come into general use. These are the improved chlorination and the cyanide method. Each of them has its strong advocates; each has its peculiar advantages and disadvantages; and neither is adaptable to the treatment of every kind of ore. The present barrel-chlorination plant is extremely simple and of moderate cost; and for ores not containing large percentages of silver or copper, or not having a lime gangue, the process is an ideal one. The

precipitated gold is clean and free from other metals, and consequently requires little after-manipulation; and regular yields of 92 to 94 per cent. of the assay-value of the ore treated are not above the normal. The disadvantages are, that the ore requires a most thorough dead roasting before it can be treated economically, and that any silver contained in the ore is lost, or must be recovered by some subsequent treatment.

The cyanide process can be worked with fairly successful results on some unroasted ores, and on some ores that are much more complex than those adapted to chlorination, but it has the disadvantage that the extraction of gold rarely reaches in practice over 80 per cent. of the assay-value, and indeed, when worked in a large scale, seldom exceeds 70 per cent., unless the ore is subjected to the same previous roasting as is necessary for chlorination. One great objection to the cyanide process is the bulk of the precipitates and the necessity for special appliances for their after-treatment. The use of zinc shavings as the precipitant requires that the zinc shall be distilled off before the gold can be melted and refined; and considerable loss occurs in this after-treatment, even under the best management. If some sure method of precipitating the gold from the cyanide solution, so that it would be free from the contamination of the zinc shavings, were devised, and a comparatively pure precipitate could be obtained without material increase in the cost of operating, then most probably the cyanide process would come as regularly into use, in this country, as the amalgamation process has done for free gold-ores. The electrolytic deposition of the gold is already being applied to the cyanide solutions in the South African mines, under the Siemens-Halske patents; but, so far, the results have not been financially more successful than those of the old McArthur-Forrest method. In fact, I find it generally reported that the yield is seldom over 70 per cent. of the value of the solution. An improvement now being tested, which is the invention of Prof. J. Richards of Lehigh University, promises, so far as laboratory-tests have shown, to overcome all of these objections to the cyanide process, besides having the further great advantage of recovering the potassium cyanide used, in condition for further use, and thus cutting down one of the chief items of cost of the whole process. It is too soon yet to predict what the practical results of these tests

will be, but should they prove successful, there would seem to be no end to the extension of the system through the low-grade mines of the South and West.

Silver and Lead.—These two metals being so intimately connected in metallurgy, they may best be considered together. Fewer changes in actual processes are to be noted for these metals than for most others, the reason probably being that when they occur together, as in most cases, nothing is simpler than blast-furnace smelting for their reduction; and when they do not occur together in nature it is generally feasible to obtain the missing one and mix them at the furnace. Where fuel is not procurable and the distance to a smelting centre is too great for the silver-value of an ore to bear transporting to it, then its treatment by itself becomes necessary, and the improvements in methods for this work have been very marked indeed, especially in the line of more rapid and thorough means for roasting and chloridizing the ores to be afterwards amalgamated, or treated by the "Russell" process, or some other method of lixiviation.

The progress during the last few years in the smelting of lead- and silver-ores has been mainly, as in the iron-furnaces, in improvement of construction, better knowledge of the chemical requirements for the constitution of the ore-charges, greater care in the choice and preparation of the fuel used, and more perfect arrangements for the collection of flue-products. The competition among the great number of smelting works erected during the last decade has so raised the prices they have to pay to the miners for ores, that the incentive to find more economical methods, and practice greater economy in the methods already known, has been particularly strong, and the results are correspondingly remarkable.

Aluminum.—Almost the entire metallurgy of aluminum, as it is practiced to-day, is the development of the last ten or twelve years. It was, I believe, at the Halifax meeting of the Institute in 1885, that Dr. T. Sterry Hunt called our attention for the first time to the electric furnace, which had been devised and tried experimentally by the Messrs. Cowles, at their Lockport works, shortly before that time. Previously, practically all the aluminum of commerce had been supplied from France, where it was made under the old Deville method of

reducing aluminum chloride by the aid of metallic sodium. It was selling in this country at a price very little below one dollar per ounce Troy. Castners' improved process for the manufacture of sodium effected such a radical economy in the price of this reducing agent as to enable the Aluminum Company of France to lower their price to seven dollars a pound; but it was not until 1887 that the Mabery and Cowles process of direct reduction in the electric furnace had progressed far enough to be able to produce aluminum at a price sufficiently low to drive out the foreign competition; and it is since that time that the phenomenal strides have been made which permit this metal to be profitably sold to-day in large quantities at about thirty cents a pound. While it is true that the Cowles process of direct reduction in the electric furnace from the oxides was the real starting-point of the movement towards the magnificent results obtained in the aluminum industry in this country, these results have only been made possible by the further invention of Mr. Charles M. Hall, now operated by the Pittsburgh Reduction Company. In this process alumina, held in solution in a bath of molten fluoride, is reduced by electrolysis without the decomposition of the bath, thus securing practically a continuous process, which furnishes a product so nearly pure as to require a simple remelting only before putting it on the market. While the extension of the use of this metal has been rapid during the last few years, the more precise knowledge of its characteristics thus acquired has very thoroughly torn away the veil of mist through which it had been studied and considered, and has resulted in considerable disappointment. As a metal by itself, it is probably not destined ever to supplant iron or steel in structural work to any extent, even if further improvements in its manufacture should enable it to be produced at equal prices. Its comparative low tensile strength, added to the extremely narrow limits of heat between which it must be worked, to insure homogeneous results, will always make it too treacherous for this purpose. On the other hand, the results already obtained with the alloys of this metal show wonderful possibilities in the future, if a reduction in the cost of their production, proportionate to what has taken place since 1890, should continue during the next few years. So far, the alloy which seems to

give the best present results and the greatest promise for the future is that with nickel, which metal has also, during the last decade, stepped out from the ranks of the rarer metals and become one of those growing into general use. The addition of a few per cent. only of nickel to aluminum greatly enhances the strength, as well as the toughness, of the metal, adding also to its brilliancy, while at the same time but slightly affecting its specific gravity.

Nickel.—As observed above, nickel in its pure state can now be classed among the commercial metals, being furnished at a price not exceeding thirty-five cents a pound. The processes used are still kept more or less secret by the manufacturers, but they are all electrolytic in character. So far the chief application of this metal has been as an alloy for copper and steel; but now that its production in a pure state and on a commercial scale has been made possible, I do not hesitate to predict a very rapid extension of its uses in the arts. Its great tensile strength, equal nearly to that of steel, together with its great ductility and its remarkably high electrical resistance, are all characteristics of general utility. Besides which, its resistance to atmospheric oxidizing agencies is almost absolute; and very few acids or alkaline solutions have any corroding effect on it.

Electrical Improvement.—In all of the lines of progress sketched above, it is evident that the principal agency which has enabled us to make such advances has been electricity. With one or two exceptions, every improvement in mining or metallurgy during the last decade is connected more or less closely with electrical development, and has been made possible by that medium. The enormous productions in our steel-works are largely due to the facility with which light and power are brought to focus at just the points where most needed. Compare the older Bessemer and open-hearth plants with the new ones of the present day. The old ones present the appearance almost of the interior of an ant-hill, so crowded are they with men pulling and tugging with levers or tongs, shouting instructions to engineers and crane-men, amid the deafening roar of steam at high pressure escaping from innumerable small or large engines. In the new mill, nearly all this is done away with, and is replaced with the almost noiseless motor, operated

almost without manual effort by the man in charge, who, placed comfortably out of the reach of the intense heat of the material or the furnaces he is operating, can overlook the whole operation, and work with a precision which was impossible under the old system. The members of the Institute will see on their visit to the works of the Illinois Steel Company, in the immense open-hearth plant of that company, one of the grandest exemplifications in the world of this application of electrically-controlled machinery, both for charging the furnaces and for handling the immense fifty-ton ladles into which the contents of the furnaces are discharged.

It is through the use of electrolysis that it has been possible to separate gold, silver and copper so economically in the Western mattes, and to produce rarer metals, such as aluminum and nickel, at a figure hardly conceived ten years ago. Altogether, this agency has been for the past decade, and will probably continue to be for the next, the main factor of progress.

There are very many other improvements I should like to mention, but for which time would be too short at this meeting. I will conclude, therefore, by merely expressing the hope that the next decade of prosperity, which now lies before us, will be fraught with as many improvements and advances in the different branches of our profession as the last one has been.

Investigations of Water-Supply.

BY F. H. NEWELL, HYDROGRAPHER UNITED STATES GEOLOGICAL SURVEY, WASHINGTON, D. C.

(Lake Superior Meeting, July, 1897.)

At the Glen Summit meeting of the Institute, held in October, 1891, the writer presented a paper on the results of stream-measurements of the United States Geological Survey, reviewing briefly the data obtained at that time.* Since then the investigations have been steadily continued and enlarged, so that at the present time about 150 river-stations are main-

* *Trans.*, xx., 547.

tained in various parts of the United States, this work being under the charge of Mr. Arthur P. Davis. At these places observations of the height of water are kept day by day, and occasional measurements are made, such that it is possible to compute the daily flow, and from this the average, the maximum, and the minimum discharge for months and years. These river-stations are situated not only upon streams within the great public land area of the United States, but, to a considerable extent, have been scattered along the Appalachians, where the results have importance in connection with plans for the development of water-powers. With the general employment of electrical means of transmission has come a demand for information concerning not only the localities where power-plants can be established, but also the volume of flow and its fluctuations.

The duration of low water and the quantity then available have special importance in such enterprises; for financial success is largely dependent upon ability to furnish a minimum power every day in the year. A few days of extreme low water habitually prevailing each year or every two or three years may be sufficient to condemn an otherwise excellent location, as it is often impracticable to provide an auxiliary steam-plant. In the Appalachian area several excellent projects have been abandoned, because of the lack of definite information on these points, and the fear that streams might shrink during summer —a fear, which, as shown by later observations, has not been well grounded.

The Division of Hydrography of the Geological Survey is conducting not only measurements of surface-streams and occasional surveys of reservoir-sites, but also examinations with regard to the amount and quality of underground waters which can be reached by ordinary or deep wells. Artesian conditions are studied in various parts of the country, and all available facts are brought together, in the hope of publishing definite information concerning the practicability of obtaining supplies of water at known depths, or at least of showing that the chances of success are great or small. This work is not as directly of an engineering character as that of stream-measurement, but involves a thorough knowledge of the geologic arrangement of the country and of the structure of the rocks, as

regards their capacity to retain and transmit water. The greater part of the work has been done within the Great Plains area, where well-water has its greatest value by reason of the absence of living streams. There, it may be said, the only mineral resource is water, and all land-values rest to a considerable extent upon the ability to procure this in considerable quantities and of good quality.

The following description of the field-work carried on in various parts of the country is arranged in general geographic order. First comes the humid region, including the eastern half of the country, or the States lying to the east of about the 97th meridian; next, the sub-humid region, which includes the States taken in order from North Dakota to Texas on the south; and finally the arid region, under which heading are included the remaining States, with the two Territories of Arizona and New Mexico.

Humid Region.

New England and New York.—The hydrographic data for the rivers of New England have been brought together by Prof. Dwight Porter, who has prepared a report upon the water-powers of this district and discussed the figures obtainable regarding river-flow. This material forms the basis for systematic examinations of the variations in discharge of some of the more important streams of this area. Facts regarding the water-power of New York are also being collected, many of these being had from investigations made primarily for the State Engineer and Surveyor, and supplemented by later research. Most of these figures have been obtained through the proprietors of water-power plants and are based upon weir-computations of overflow-dams or on estimates of the capacities of water-wheels.

Pennsylvania and Maryland.—Systematic measurements have been gradually extended northerly from the Potomac into these States. A station has been located at Harrisburg on the Susquehanna river and observations are also being carried on at points on Octoraro creek, Patapsco and Patuxent rivers and Antietam creek, as well as at points in Maryland on the Potomac. The work in the latter State is conducted in co-operation with Prof. W. B. Clark, State Geologist.

Virginia and West Virginia.—Measurements of the Potomac

river and its tributaries and also of some of the streams forming part of the Ohio drainage have been continued under the supervision of Prof. D. C. Humphreys, of Lexington, Va. In connection with the systematic work, he has continued at intervals a general reconnaissance of localities where water-powers may be developed.

North and South Carolina.—In co-operation with Prof. J. A. Holmes, State Geologist, the river-stations have been maintained in these States, and the number has been slightly increased. The field-work has been carried on mainly by his assistant, Mr. E. W. Myers.

Georgia, Alabama and Tennessee.—The operations in Georgia, under the direction of Prof. B. M. Hall, of Atlanta, have been gradually extended to streams rising in the southern end of the Appalachian area and flowing southerly through Alabama and westerly into Tennessee. The work in Alabama has been carried on in co-operation with Prof. Eugene A. Smith, State Geologist, who has also collected data concerning the waters obtained by wells.

Ohio, Indiana and Illinois.—The hydrographic data obtained by Mr. Frank Leverett in glacial studies extending across Ohio and Indiana, have been compiled by him, being rounded out by later observations. These facts pertain mainly to the well-waters of these States. The results are being published in Part IV. of the *Eighteenth Annual Report* of the United States Geological Survey. He has also continued the compilation of data regarding the wells of Illinois, this material being supplementary to his paper published in the *Seventeenth Annual Report.* In addition, Prof. Edward Orton, State Geologist of Ohio, has brought together a considerable body of facts relating to the water-resources from the deeper wells of that State, this material being obtained in part in connection with his investigations of the occurrence of oil and gas.

Michigan.—Prof. Alfred C. Lane, assistant State Geologist, has begun the collection of facts concerning well-waters through correspondence by schedules, with the intention of supplementing this information by occasional field-work.

Sub-humid Region.

The sub-humid region is for convenience regarded as including the greater parts of the States of North and South Dakota,

Nebraska, Kansas, Texas and the Territory of Oklahoma. The principal work carried on in these States has been in connection with artesian investigations, but a considerable number of stream-measurements have also been made.

North and South Dakota.—Prof. Earle J. Babcock, of Grand Forks, North Dakota, has carried on a reconnaissance of the area in the vicinity of Devils Lake, North Dakota, and to the north. The results of his work have been prepared for publication as one of the series of "Water Supply and Irrigation Papers." Mr. N. H. Darton, assistant geologist, has conducted some field-work in South Dakota, and has prepared a paper now being published, giving his conclusions; this being in a certain sense a continuation of his paper in the *Seventeenth Annual Report*, now in print.

Nebraska.—The largest single investigation undertaken during the year has been the systematic work of Mr. Darton in his thorough examination of the country from the vicinity of Lincoln westward along the Platte valley to the vicinity of Lexington. In this work he was assisted by Mr. Joseph Macfarland, a young man of considerable ability, whose death at the end of the field-season deprived this survey of one of its most energetic younger investigators. The conclusions reached by Mr. Darton have been embodied in a paper to be printed as one of the series on water supply and irrigation. Prof. Edwin H. Barbour, acting State geologist, has co-operated in a general study of the well-waters of the State, and has prepared a paper upon the utilization of these, especially through the employment of wind-mills in bringing the water to the surface. The river-measurements in this State have been carried on under the direction of Prof. O. V. P. Stout, with the exception of the measurements at North Platte, in charge of Mr. Charles P. Ross.

Kansas.—Prof. Erasmus Haworth, of the Kansas University Geological Survey, has prepared a report upon the water-conditions of the area south of the Arkansas river, in the vicinity of Dodge and Garden. This has been published as No. 6 of the "Water Supply and Irrigation Papers." Mr. Willard D. Johnson has spent the greater part of the year in western Kansas, working with various investigators and bringing together information regarding the behavior of water underground.

Prof. E. C. Murphy, of Lawrence, has conducted a series of

experiments on the efficiency of wind-mills in raising water for irrigation, taking the mills in actual operation in the vicinity of Garden, Kansas, and has prepared for publication, as Paper No. 8 of the above series, a description of his tests. A similar line of investigation has been pursued by Prof. O. P. Hood, at Manhattan, upon the efficiency of various pumps commonly employed in raising water for irrigation. The stream-measurements of the western part of the State have been under the charge of Mr. W. G. Russell, while those in the eastern end, in the vicinity of Lawrence, have been carried on by Prof. E. C. Murphy.

Texas.—A report upon the utilization of water in irrigation in portions of Texas has been prepared under the direction of Prof. J. H. Connell, of Bryan, by Mr. William Ferguson Hutson. A special investigation has also been made by Prof. Thomas U. Taylor, of Austin, on the rate of silting of Lake McDonald, the reservoir formed by the Austin dam; data of this character being of great importance in estimates of available water from storage-projects.

Arid Region.

In discussing the hydrographic work of the arid region, the alphabetic instead of geographic order is adopted, as being more convenient for general reference. The principal work here has been in the measurements of discharge of streams rising in the high mountains and flowing out upon the broader valleys. The results of these measurements have been discussed in a number of papers in connection with the utilization of the waters in irrigation.

Arizona.—In this Territory observations of water-supply have been continued at the Buttes, above Florence, to obtain the amount of water which might be held by a suitable storage-reservoir at that point. Computations of discharge have also been made for the Verde and Salt rivers near their junction.

California.—The river-measurements in this State have been continued by Mr. J. B. Lippincott, who has also prepared a paper upon the water-supply of the San Bernardino valley. Mr. C. E. Grunsky, of San Francisco, has written a report upon irrigation in the southern or upper end of the San Joaquin valley, and has brought together material for a description of the area further north. Mr. James D. Schuyler, of Los Angeles,

has prepared a thoroughly illustrated discussion of the principal storage-dams and reservoirs constructed for purpose of irrigation, this being printed as one of the papers in Part IV. of the *Eighteenth Annual Report.*

Colorado.—The State engineer of Colorado has continued his co-operation in the measurement of the discharge of rivers of this State, assistance being given by Mr. F. Cogswell, deputy State engineer. Mr. David Boyd, of Greeley, has prepared a paper on the development of irrigation and of the Colorado system of legislation, founded largely on the experience of the Union Colony.

Idaho.—Mr. F. J. Mills, State engineer of Idaho, has been of assistance in continuing the river-measurements in this State, the work being carried on during the past year mainly by Mr. Grant H. Nickerson; data regarding the Bruneau river have also been obtained from Mr. Andrew J. Wiley, of Grand View.

Montana.—The river-stations, which are mainly in or adjacent to the Gallatin valley, have been maintained by Prof. Frank Beach, of Bozeman, assisted by Mr. Roe Emery.

Nevada.—Mr. L. H. Taylor, of Battle Mountain, has continued the field-work at various points along the Humboldt river and on its tributary, Rock creek.

New Mexico.—The work along the Rio Grande has been continued under the charge of Mr. P. E. Harroun, except at El Paso, where, through the request of Gen. Anson Mills, the station has been placed under the charge of Mr. W. W. Follett, civil engineer for the Mexican Boundary Commission.

Oregon.—The river-stations in the eastern end of this State have been under the charge of Mr. Grant H. Nickerson; the work being done in connection with that in Idaho. In the northeastern part of the State the river-measurements have been made by Mr. Cyrus C. Babb.

Utah.—Prof. Samuel Fortier, of Logan, has continued work at the river-stations, with the assistance of Mr. William Dougall, and also has conducted seepage-measurements, being aided in the latter by county authorities and others; the results being of immediate practical benefit to the inhabitants of the northern part of the State. The results of this latter work are given in Paper No. 7 of the "Water Supply and Irrigation" series.

Washington.—Mr. Cyrus C. Babb, assistant hydrographer,

has devoted a considerable part of his time to measurements in this State, having also inspected the operations of resident hydrographers in other localities.

Wyoming.—Prof. Elwood Mead, State engineer, has continued to render assistance to the survey; the work of stream-measurement being carried on by his assistant, Mr. C. T. Johnson. Aid has also been rendered by Mr. Edward Gillette, at Sheridan, Wyoming.

List of Rivers Measured.

The following list gives in geographic order, by drainage-basins, the localities at which systematic measurements or computations of discharge have been made. At most of these the work has been done by hydrographers of this survey, while at a few the data have been obtained from engineers, being rounded out by additional measurements or by computation.

List of Rivers Measured in 1896–97.

River.	State.
Chesapeake watershed:	
Susquehanna river at Harrisburg,	Pennsylvania.
Octoraro creek at Rowlandville,	Maryland.
Patapsco river at Woodstock,	"
Patuxent river at Laurel,	"
Potomac basin:	
South Branch of Potomac river at Springfield,	West Virginia.
Potomac river at Cumberland,	Maryland.
Antietam river at Sharpsburg,	"
North river at Port Republic,	Virginia.
South river at Port Republic,	"
Shenandoah river at Millville,	West Virginia.
Potomac river at Point of Rocks,	Maryland.
Monocacy river at Frederick,	"
James River basin:	
North Fork of James river at Glasgow,	Virginia.
James river at Buchanan,	"
South Atlantic watershed:	
Roanoke river at Roanoke,	Virginia.
Dan river at Clarksville,	"
Staunton river at Clarksville,	"

River.	State.
Roanoke river at Neal,	North Carolina.
Tar river at Tarboro,	"
Neuse river at Selma,	"
Cape Fear river at Fayetteville,	"
Yadkin river at Salisbury,	"
Yadkin river at Norwood,	"
Catawba river at Rock Hill,	South Carolina.
Catawba river at Catawba,	"
Broad river at Gaffney,	"
Broad river at Alston,	"
Saluda river at Waterloo,	"
Savannah river at Calhoun Falls,	"
Oconee river at Carey,	Georgia.
Ocmulgee river at Macon,	"
Gulf of Mexico watershed:	
Chattahoochee river at Oakdale,	Georgia.
Chattahoochee river at West Point,	"
Etowah river at Canton,	"
Coosawattee river at Carters,	"
Oostanaula river at Resaca,	"
Coosa river at Riverside,	Alabama.
Coosa river at Lock No. 5,	"
Black Warrior river at Tuscaloosa,	"
Ohio River tributaries:	
Greenbrier river at Alderson,	West Virginia.
New river at Fayette,	"
French Broad river at Asheville,	North Carolina.
Tuckaseigee river at Bryson,	"
Little Tennessee river at Judson,	"
Hiwassee river at Murphy,	"
Tennessee river at Chattanooga,	Tennessee.
Upper Missouri basin:	
Missouri river at Townsend,	Montana.
West Gallatin river at Salesville,	"
Middle creek at Bozeman,	"
Gallatin river at Logan,	"
Madison river at Redbluff,	"
Jefferson river at Sappington,	"
Yellowstone river at Livingston,	"

River.	State.
Shoshone river at Lovell,	Wyoming.
Grey Bull river at Meeteetse,	"
Little Goose creek at Sheridan,	"
Big Goose creek at Sheridan,	"
Clear creek at Buffalo,	"
Platte basin:	
Laramie river at Woods Landing,	Wyoming.
Laramie river at Uva,	"
North Platte river at Orin Junction,	"
North Platte river at Camp Clarke,	Nebraska.
North Platte river at North Platte,	"
South Platte river at Deansbury,	Colorado.
South Platte river at Denver,	"
South Platte river at Orchard,	"
North Loup river at St. Paul,	Nebraska.
Middle Loup river at St. Paul,	"
Loup river at Columbus,	"
Platte river at Columbus,	"
Elkhorn river at Norfolk,	"
Kansas basin:	
North Fork of Republican river at Haigler,	Nebraska.
Frenchman river at Wauneta,	"
Frenchman river at Palisade,	"
Republican river at Superior,	"
Republican river at Junction City,	Kansas.
Solomon river at Beloit and Niles,	"
Saline river at Beverly and Salina,	"
Smoky Hill river at Ellsworth,	"
Blue river at Manhattan,	"
Kansas river at Lawrence,	"
Arkansas basin:	
Arkansas river at Granite,	Colorado.
Arkansas river at Salida,	"
Arkansas river at Canyon,	"
Arkansas river at Pueblo,	"
Purgatoire river at Trinidad,	"
Arkansas river at Hutchinson,	Kansas.
Verdigris river at Liberty,	"
Neosho river at Iola,	"

River.	State.
Rio Grande basin:	
Rio Grande at Del Norte,	Colorado.
Rio Grande at Embudo,	New Mexico.
Chama river at Abiquiu,	"
Rio Grande at Rio Grande,	"
Rio Grande at San Marcial,	"
Rio Grande at El Paso,	Texas.
Colorado basin:	
Grand river at Grand Junction,	Colorado.
Dolores river at Dolores,	"
San Miguel river at Fall Creek,	"
Uncompahgre river at Fort Crawford,	"
Blacks Fork river at Granger,	Wyoming.
Green river at Green River,	"
Green river at Blake,	Utah.
San Juan river at Arboles,	Colorado.
Piedras river at Arboles,	"
Animas river at Durango,	"
Gila river at Buttes,	Arizona.
Salt river at Mt. McDowell,	"
Verde river at Mt. McDowell,	"
Interior basin in Nevada:	
Humboldt river at Elko,	Nevada,
Humboldt river at Battle Mountain,	"
Humboldt river at Golconda,	"
Humboldt river at Oreana,	"
Rock creek at Battle Mountain,	"
South Fork of Humboldt river at Mason's Ranch,	"
Interior basin in Utah and Idaho:	
Logan river at Logan,	Utah,
Bear river at Collinston,	"
Ogden river at Eden,	"
Weber river at Uinta,	"
Provo river at Provo,	"
Utah lake,	"
Columbia basin:	
Snake river at Montgomery Ferry,	Idaho.
Malad river at Taponis,	"
Little Wood river at Taponis,	"

River.	State.
Bruneau river at Grand View,	Idaho.
Boise river at Boise,	"
Owyhee river at Nyssa,	Oregon.
Malheur river at Vale,	"
Payette river at Payette,	Idaho.
Weiser river at Weiser,	"
Yakima river at Selah,	Washington.
Yakima river at Union Gap,	"
Yakima river at Kiona,	"
Spokane river at Spokane,	"
Umatilla river at Gibbon,	Oregon.
San Francisco Bay drainage:	
Sacramento river at Redbluff,	California.
Sacramento river at Jellys Ferry,	"
Stanislaus river at Oakdale,	"
Tuolumne river at La Grange,	"
Tuolumne river at Modesto,	"
San Joaquin river at Herndon,	"
Kings river at Red Mountain,	"
Kings river at Kingsburg,	"
Kern river near Bakersfield,	"
Small "Lost Creeks" in southern California:	
Little Rock creek at Palmdale,	California.
San Gabriel river at Azusa,	"
Santa Ana river at Warm Springs,	"

Reports.

The results of the operations during the past year are shown in part in Part IV. of the *Eighteenth Annual Report* of the Survey, which has been devoted to hydrography.

In addition to the annual report, the Division of Hydrography now has a separate series of publications, referred to above, entitled, "Water Supply and Irrigation Papers." Of these about twenty-five have been prepared from the operations of the year. Six have already been printed, and others are in the hands of the printer or are awaiting publication. The following list gives the names of these, the order of papers after No. 13 not having been determined as yet.

Water-Supply and Irrigation Papers.

1. Pumping Water for Irrigation, by Herbert M. Wilson, 1896.
2. Irrigation near Phœnix, Arizona, by Arthur P. Davis, 1897.
3. Sewage Irrigation, by George W. Rafter, 1897.
4. A Reconnaissance in Southeastern Washington, by Israel C. Russell, 1897.
5. Irrigation Practice on the Great Plains, by E. B. Cowgill, 1897.
6. Underground Waters of Southwestern Kansas, by Erasmus Haworth, 1897.
7. Seepage-Waters of Northern Utah, by Samuel Fortier.
8. Windmills for Irrigation, by E. C. Murphy.
9. Irrigation near Greeley, Colorado, by David Boyd.
10. Irrigation in Mesilla Valley, by F. C. Barker.
11. River-Heights for 1896, by Arthur P. Davis.
12. Water-Resources of Southeastern Nebraska, by Nelson Horatio Darton.
13. Water-Resources of the Devils Lake Region, North Dakota, by Earle J. Babcock.

Notes on Six Months' Working of Dover Furnace, Canal Dover, Ohio.

BY ARNOLD K. REESE, SPARROWS POINT, MD.

(Lake Superior Meeting, July, 1897.)

It is not the purpose of the writer to set forth in these notes anything new or surprising in blast-furnace practice, but simply to lay before the Institute the somewhat unusual results obtained during a period of six consecutive months, ending April 30, 1896, at the Dover furnace of the Penn Iron and Coal Co., and to give, with these results, the conditions under which they were obtained, and all figures pertaining to that period.

This plant was entirely rebuilt, on thoroughly modern and improved plans, in the spring of 1895, by Frank C. Roberts & Co., of Philadelphia.

A novel feature in the construction is the double or Y-shaped down-comer, so arranged as to take the waste gases from two openings at the top, each having an area equal to that of the single part where the two branches join, at a point below, about midway between the top and the dust-catcher.

This construction is, I believe, in itself novel,* though its purpose may not be so. The area of the openings for the discharge of waste gases is increased, and consequently their velocity is reduced, which reduction in velocity tends to lessen the amount of fine materials, or flue-dust, carried over. This is an important point, as furnace-men well know, in regions where very fine ores are used, a large percentage of the dust consisting of these ores. Analyses of such flue-dust show at times as much as 56 per cent. of metallic iron.

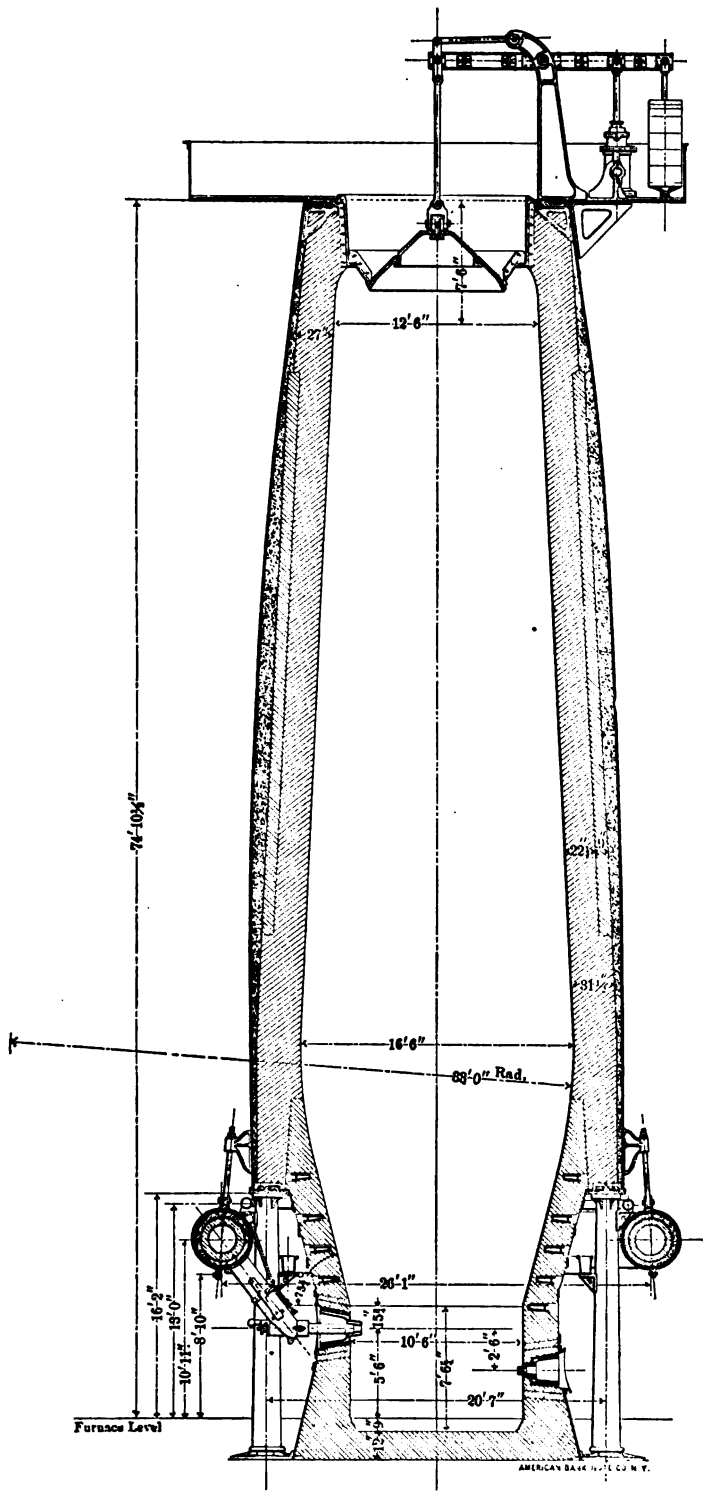
As to the efficiency of this device I can draw no conclusions, having no figures from similar mixtures with the ordinary single down-take, but others who have such figures may find it interesting to compare them with those I give below.

With the exception of a very few short periods, when coke-shipments were light and it became necessary to run to our stock-pile and fork coke, all fuel used at this furnace was drawn from bunkers, into which it was thrown from the cars, without screening of any kind, so that all dirt and small coke went into the furnace. I found that during the periods when forked coke was working on the furnace, the iron was invariably hotter and the silicon higher, and that the forked coke would carry from 300 to 400 pounds more burden per charge than the bunker-coke. Every barrow of coke was weighed, except during periods of heavy or long-continued rains.

The Dover plant consists of one stack 75 feet high, 16 feet 6 inches in diameter of bosh, 10 feet 6 inches in diameter of hearth and 12 feet 6 inches in diameter at stock-line. (For other dimensions, see Fig. 1.) There are three Cowper hot-blast stoves with Roberts brick, 65 feet high by 16 feet in diameter; six 150-H. P. Stirling water-tube boilers in three

* I have since learned that a similar device is used at the Carrie furnace near Pittsburgh, though with somewhat different arrangement, the branches at the top being horizontal and the down-comer vertical. At the Dover furnace, both are inclined, and the great accumulation of dust which must take place in the horizontal branches is avoided.

FIG. 1.



Elevation Dover Furnace.

batteries; two old double return-flue boilers (used exclusively for the pumps); one new improved vertical Allis blowing-engine, 42 inches by 84 inches by 60 inches, with Reynolds positive-closing discharge-valves; and one old Weimer blowing-engine (out of use for extensive repairs and remodeling), 42 inches by 84 inches by 48 inches.

During the entire period treated of in these notes the furnace was blown with the Allis engine alone, with a maximum speed of 45 revolutions per minute, giving a piston-delivery of 17,268 cubic feet. (It may be needless to say that the Allis engine was run at its maximum speed constantly.)

The fuel was standard Connellsville coke and was throughout of uniformly good character.

Average of Eighteen Analyses of Coke.

	Per cent.
Volatile matter,	3.65
Ash,	12.42
Carbon,	83.12
Silica,	4.43
Alumina,	2.52
Sulphur,	0.71

The limestone was of a quality which could not be called first-class, being rather high (and varying considerably) in silica, though it was no worse than is used in many other places.

Average Analysis of Eighteen Unselected Samples from Pile, taken during February and March.

	Per cent.
Silica,	5.44
Alumina,	2.42
Lime,	50.30
Magnesia,	0.46

Of these eighteen samples the highest silica was 8.90 per cent. and the lowest was 3.70 per cent. No serious trouble was experienced from this source, though the more or less frequent sudden changes in the cinder may have been, and probably were, due to it.

The ores were undoubtedly a choice selection; few, if any, existing in the Mesabi range better than the Franklin, both as to chemical composition and as to physical properties. The

Aurora and Chapin are both good ores, while the Rex, Atlantic and Tilden, of which small quantities were used at times, though leaner than the former, are good working-ores. Varying quantities of Marquette, mill-cinder (as silica introducers) and mill-scale were also used.

The following table gives the average analysis of ores.

TABLE I.—*Average Analysis of Ores.*

	Moisture.	Fe.	SiO_2.	P.	S.	CaO.	Al_2O_3.	MgO.	Mn.
Franklin.....	6.88	64.44	2.85	.033	.012	0.18	0.67	0.060	0.55
Aurora.....	9.00	63.10	3.90	.029	.010	0.10	0.96	0.050	0.40
Chapin.....	4.88	61.25	3.75	.070	.012	1.15	0.98	2.80	0.55
Rex.....	6.15	57.11	6.50	.066	.022	1.38	1.61	3.00	1.08
Atlantic.....	9.38	62.75	4.90	.037	.013	2.08	1.18	0.129	1.28
Tilden.....	14.76	63.39	4.31	.038	.010	0.12	1.00	0.13	1.18
Marquette.....	6.55	43.18	33.00	.058	.027	0.23	1.35	0.18	0.43
Mill-cinder.....	.....	55.70	23.95	.162	.597	.....	3.70	.....	.....
Mill-scale.....	.....	.....	1.32	.015	.....	.....	.....	.....	.....

All of these are well known to users of Lake ores and further comment on them is not necessary.

Such a mixture was aimed at as would give 6 per cent. of SiO_2 in the furnace, exclusive of that introduced with stone and fuel. It was considered more economical to obtain silicon in the iron from the presence of a higher percentage of SiO_2 in the burden, by the use of more lean ore high in SiO_2 (Marquette), than by saving at this point and obtaining the silicon at the expense of fuel; since the smaller the amount of SiO_2 in the burden, the hotter must be the furnace to reduce a given quantity of SiO_2 to silicon.

The general policy as to the cinder was to carry the most acid cinder with which it was possible to keep the sulphur below the limit, 0.05 per cent.

The fuel unit throughout consisted of 5000 pounds of coke. Both coke- and ore-scales were tested weekly.

For convenience in reading, the figures showing general results in monthly averages are given in tabulated form in Table II.

Attention may be called to the fact that in the month of lowest fuel consumption (March) the greatest total output was obtained, with the smallest amount of re-melted iron.

TABLE II.—*Conditions of Production.*

Month.	Ratio of Ore to Coke.	Production from Ores.	Iron Remelted.	Total Output.	Daily Average.	Burden Yield.	Fuel Consumption.	Fuel-Ratio.
		Tons.*	Tons.*	Tons.*	Tons.*	Per ct.	Pounds.	
November	2.16	5740.1	340?	6080.1	202.6	59.1	1666	0.734
December	2.14	6018.0	136	6154.0	198.7	60.7	1683	0.741
January	2.19	6088.2	291	6379.2	205.78	58.7	1655	0.729
February	2.247	6043.7	157	6200.7	213.8	59.0	1645	0.72
March	2.28	6533.2	127	6660.2	214.8	58.5	1645	0.72
April	2.20	6017.2	201	6218.2	207.27	58.2	1689	0.74
Total	.....	36,440.4	1252	37,692.4	.....	.....	.....	.....
Average	2.203	.....	.....	.....	207.09	58.99	1664.3	0.733
Greatest variation	0.14	.....	.....	.....	16.10	2.5	44.	0.02

Month.	Extra Time Lost.		Stone per Ton of Iron Produced.	Blast-Temperature.	Blast-Pressure.	Top-Temperature.	Piston Displace. Cubic Feet Blast per Minute.	Per Cent. No. 1 Bessemer of Output.	Per Cent. No. 1 Bessemer of Production.
	Hrs.	Min.	Pounds	Fahr.	Pounds	Fahr.	Cu. feet.		
November	12	36	753.6	1050°	6.75	480°	17268	99.38	99.34
December	30	2	780.5	1080°	6.5	450°	17268	96.27	96.19
January	16	30	811.6	1050°	6.5	375°	17268	96.65	96.49
February	13	20	797.8	1150°	7.	425°	17268	100.00	100.00
March	3	40	759.1	1150°	7.	425°	17268	98.34	98.29
April	1	56	722.7	1170°	6.5	450°	17268	98.42	98.37
Total	78	4	.....	.....	.....	.....	.....	.....	.....
Average	.....		770.99	1108.3°	6.71	434°	17268	97.895	97.823
Greatest variation	.....		89.00	.....	0.5	105°	.....	3.73	3.81

The regular working of the furnace is shown in the line of greatest variation.

Table III. gives the chemical analyses of products and Table IV. some further figures, furnishing additional means of comparison.

All of the chemical analyses were made by Mr. David T. Croxton, then chemist of the works.

* Ton = 2268 pounds.

TABLE III.—*Analyses of Products.*

Month.	CINDER.				IRON.			GAS.		
	SiO_2.	CaO.	Al_2O_3.	MgO.	Si.	S.	P.	CO.	CO_2.	$\frac{CO}{CO_2}$.
November	34.50	43.00	16.00	5.00	1.25	.030	.078	.....	.....	.....
December	34.50	42.00	14.50	5.00	1.40	.035	.078	.....	.....	.....
January	34.24	43.38	15.24	4.96	1.25	.035	.080	.....	.....	.....
February	33.80	43.28	15.29	4.29	1.10	.035	.080	.....	.....	.....
March	34.26	43.32	17.71	4.48	1.16	.038	.082	22.66	14.26	1.59
April	34.93	43.80	13.51	.....	.....	.....	.....	19.16	15.42	1.24
Average	34.37	43.13	15.37	4.74	1.23	.034	.079	19.97	15.15	1.32*

As these tables are sufficiently explicit in themselves, further remarks upon them are unnecessary, except perhaps a few words on the extremely low ratios of CO to CO_2 in the few analyses of gas taken from the furnace-top, which are far below the theoretical limit for economical work as given by Sir Lowthian Bell in his "Principles of the Manufacture of Iron and Steel."

TABLE IV.—*Blast and Capacity.*

Month.	Cubic Feet Air per Ton Iron.		Cubic Feet Air per Pound Coke Consumed.	Tons of Iron per Square Foot Hearth Area per 24 Hours.		Tons of Iron per 1000 Cubic Feet Cubic-Capacity of Furnace per 24 Hours.	
	(1). Total Iron.	(2). Deducting Remelt.		(1).	(2).	(1).	(2).
November	112,662	121,079	67.51	2.500	2.359	16.74	15.81
December	114,534	117,123	68.11	2.508	2.452	16.80	16.43
January	112,840	118,235	68.20	2.544	2.428	17.05	16.28
February	110,226	111,435	67.01	2.604	2.537	17.45	17.00
March	110,854	113,009	67.27	2.590	2.541	17.36	17.03
April	114,810	118,645	67.96	2.501	2.420	16.76	16.22
Average	112,654	116,588	67.68	2.539	2.456	17.04	16.46

These ratios would seem, however, to be in keeping with the low fuel-consumptions accompanying them, and also with the low blast-temperatures obtained from the stoves, which should have been higher with a higher ratio of CO to CO_2 in the gases, and the small amount of blast blown per minute.

* Average of 13 analyses.

I believe a gas similar to these was obtained at the Union Works of the Illinois Steel Company, as shown in the interesting paper of Mr. Whiting in Vol. XX. of the *Transactions*.*

In Table V. are given the ore-mixtures for each month.

TABLE V.—*Proportions of Ores Used.*

Ores.	November.	December.	January.	February.	March.	April.	May.
Chapin	34.2	.....	.....	.....	33.41	33.18	29.35
Chapin and Rex	.....	34.9	34.99	34.07	.....	.....	.....
Franklin	33.2	34.9	34.99	34.07	33.41	33.18	36.02
Aurora	23.8	23.25	23.32	22.71	.....	.....	.....
Aurora and Tilden	.....	.....	.....	.....	22.27	.....	.....
Atlantic and Tilden	.....	.....	.....	.....	.....	.....	24.69
Tilden	.....	.....	.....	.....	.....	22.12	.....
Marquette and Mill-cinder	8.8	6.6	6.33	6.79	7.65	7.86	9.70
Mill-scale	.....	0.35	0.37	2.36	3.26	3.66	0.24

Returning to the subject of the flue-dirt taken from the dust-catcher, I have the figures contained in Table VI., which are necessarily approximate.

They are derived from a record of the number of cart-loads of dust taken out each day, a number of these having been weighed and their average weight taken as a basis for the estimate. As the dust was cooled with water before being thrown into the carts, the figures will tend to be too high rather than too low.

From analyses of the flue-dirt, which gave, when dry, upwards of 56 per cent. of metallic iron, it may be assumed, with a considerable degree of accuracy, that the dust contained fully 90 per cent. of iron-ore.

On this assumption, the figures in the fourth column of Table VI. have been made.

Only very small quantities of dust were taken from time to time from the combustion-chambers of the stoves and boilers; and not once during twelve months was it found necessary to take off a stove for cleaning. Upon examination of the stoves after a year's run, in only one, the furthest from the furnace, was any considerable quantity of dust found.

An examination of Table VI. shows that, with an approximately constant percentage of Mesabi ore in the burden, we

* *Trans.*, XX., 280.

TABLE VI.—*Approximate Quantity of Flue-Dust Taken from Dust-Catcher.*

Month.	Weight of Dust.	Total Weight of Ore Charged.	Per Cent. of Mesabi Ore.	Per Cent. of Total Ore in Dust.
	Pounds.	Pounds.		
November	454,720	21,907,760	33.15	1.87
December.	235,200	22,202,610	34.88	0.95
January	277,760	23,203,860	34.99	1.07
February	275,520	22,921,800	34.07	1.08
March	374,080	25,014,460	33.41	1.34
April	398,720	23,131,240	33.18	1.55
Total	2,016,000	138,381,730	.....	.....
Average	.....	.....	33.947	1.31

have a very low relative percentage of flue-dust in the months of December, January and February, with a decided increase as we advance through March and April, and a like increase in November. A very simple explanation is found for this in the condition of the ores, due to the effects of the season. The lowest percentages are in those months when the ores were charged either in a frozen or very moist condition, due to winter rains, snows and frosts. The quantity of dust increases as we approach the drier months, until we reach May (usually a very dry month), when a relatively excessive quantity is found. A similar increase would undoubtedly be found in September and October, were figures for these months at hand.

Comparisons with these figures should be made by months, and climatic conditions should be taken into account.

The Influence of Lead on Rolled and Drawn Brass.

BY ERWIN S. SPERRY, BRIDGEPORT, CONN.

(Lake Superior Meeting, July, 1897.)

METALS differ widely in their behavior under the cutting-tool. Some, like iron or steel, require a slow speed and light feed, a tool shaped differently from that used for other metals,

and usually oil must be applied to the work in order to obtain a free cut. Others, like cast-iron, are worked with a speed somewhat higher, and a cut considerably heavier than that employed for iron or steel, and commonly no oil is applied, while others can not only be cut without oil, but the speed can be very high.

An example of this last class is brass, an alloy employed in the arts for purposes demanding a free-working material possessing moderate strength. The name "*brass*," however, is a term of considerable latitude. It is generally associated with a material of excellent cutting-qualities,* but the variation is great in this respect. As a rule, the lower the brass the more difficult it is to cut. As the composition approaches pure copper it becomes, like this metal, one of the most troublesome substances known to machine.

If the alloy is made from pure copper and spelter (zinc), the result is a material wholly unsuited for certain uses; its chips are long and tenacious; a slower speed must be employed in cutting, and even oil must be applied in many instances. If drilled at a high speed, the drill is liable to become drawn in and broken, and a burr is left on the reverse side which requires extra treatment to remove. If employed in an automatic machine, such as a screw-machine, the chips are liable to obstruct the mechanism. In filing, the file soon becomes clogged with particles of metal, which abrade the work in grooves difficult to obliterate.

For certain classes of work, such as spinning- or cartridge-brass, these qualities are essential; but for others, like clock-brass or screw-rod, they are almost prohibitory. Practice has decided that for the latter purpose all elements of a refractory nature shall be removed, and an alloy produced which can be worked free at a high speed. The difficulty can be partially overcome by making an alloy which will give short chips, but an alloy of this description must still possess sufficient ductility to roll and draw. The methods of making such an alloy are as follows:

1. By making the brass very high, say not below copper, 55, and zinc, 45 per cent. Such a mixture could be broken down

* The terms "*high*" and "*low*" are now accepted phraseology, and are used to designate the contents of zinc. Low brass contains less zinc than high brass.

hot, but even then would be unsuitable on account of its hardness and difficulty of rolling or drawing. The chips of the alloy are fairly short, but the speed of working would have to be slow, on account of the hardness.

2. By the addition of tin. Tin, even in small quantities, hardens brass, and gives a metal similar to that proposed in the first case. The chips are short if the content of tin is sufficient, but the hardness is too great for a high speed.

3. By the addition of lead. This method of producing free-working brass is now practiced in all the brass-manufacturing establishments in the world. How long it has been known I am unable to say. It was probably discovered by accident, as the following extract, taken from an old metallurgical work* will show. The original article is present in a contemporary work† which I was unable to consult. The translation is as follows:

"The results of observations made by M. Chaudet are that brass usually made from copper and zinc‡ oftentimes contains from 2 to 3 per cent. of lead. It even appears that this metal gives to the brass properties much sought for by turners; in fact, M Chaudet, having had occasion to analyze three samples of brass, found that two of these samples, prized for turning, but not at all suitable for hammered work, contained, in the one case, 2.86, and, in the other, 2.15 per cent. of lead, while the third, valued highly for hammered work and employed with difficulty for turning, contained only traces. M. Chaudet is certain, besides, that in combination with such a quantity of lead, brass is rendered less ductile, and assumes properties which turners desire. Lead gives to brass the quality of "shortness," and obviates the necessity of oiling the work when it is desired to prevent the metal from clogging the tool and the chips to be freed from it under its influence. Leaded brass can be sawed, slit and drilled with neatness and precision."

It appears, therefore, that if, before that time, lead was added to brass to give it such properties, the knowledge was a trade secret; but in all probability its presence in the samples investigated by M. Chaudet was an accident, for in those times both zinc and copper contained considerable lead, especially the latter, which was added purposely in the copper-refining process.

The credit of the discovery of the fact that lead gives to brass desirable cutting-qualities is undoubtedly due to M.

* *Nouveau Manuel Complet des Alliages Métalliques*, par A. Herve, Paris, 1839, page 362.

† *Manuel de l'Essayeur*, page 370.

‡ Percy's *Metallurgy*, vol. i.—*Copper, Zinc and Brass*, page 621. In contradistinction to brass made by the old calamine method, now obsolete.

Chaudet, and is a valuable one from a metallurgical standpoint, for if such a practice were not followed at the present time, methods would have to be changed at the expense of economy of production.

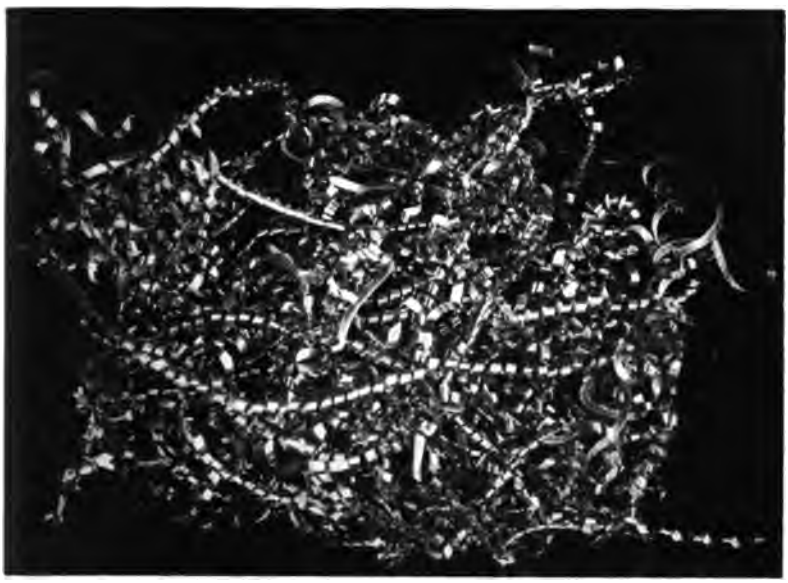
The consumption of brass containing lead is very large, and such material is known to the trade as "*leaded-brass*," of which "*clock-brass*" and "*screw- or drill-rod*" are two of the most common and largely-used varieties. The amount of lead added to such a mixture is from 1½ to 2 pounds per hundred. This amount is sufficient to give the alloy the desired properties.

Figs. 1 and 2, reproduced from photographs, will show the effect of a small amount of lead on brass. Fig. 1 shows chips turned from an alloy consisting of pure copper and zinc without lead. The chips are long and tenacious, and an alloy producing such chips is wholly unsuited for use in an automatic machine intended for working brass. Fig. 2 shows chips turned from brass of the same composition as that of Fig. 1, but with the exception that it contained 2 per cent. of lead. The chips in this case were very short, and a much higher speed could be employed in cutting. These two alloys would probably show nearly the same degree of hardness, but the cutting qualities are so dissimilar that one would hardly believe the composition to be almost identical.

Just what this action of lead on the brass is, that such a change should be produced in its physical properties, has been incompletely understood. M. Chaudet says that brass is rendered less ductile by such an addition, but it is evident that the ductility cannot be much reduced, as the alloys will roll and draw perfectly. In order to investigate this matter, and, if possible, find the effect of the lead in varying quantities, the following experiments were made:

Varying amounts of lead were added to brass of a constant composition, and the resulting alloys were tested. For the materials the purest Lake copper, Bertha zinc, and pure desilverized lead were used. The copper was melted in a clean plumbago crucible under a layer of charcoal, care being taken to avoid overheating. When melted the zinc was added, and lastly the lead; and the whole mixture afterwards thoroughly stirred with a plumbago stirrer. The molten metal was next poured into an iron mould, ¼-inch by 2 inches by 12

FIG. 1.



Brass Chips Free from Lead.

FIG. 2.

Brass Chips Containing 2 per cent. of Lead.

inches inside. The pouring was made at the lowest possible temperature, and the surfaces of the mould were coated with sperm oil. In order to make sure that the lead was thoroughly distributed through the alloy, a second stirring was given just before pouring. The surfaces of the plate were overhauled after being cast by planing, so as to obtain a clean surface before rolling. The various annealings were made with gas.

Experiment A.—Copper 60, *Zinc* 30, *and Lead* 10 *per cent.*

Melted 6 pounds of copper and then added 3 pounds of zinc, and afterwards 1 pound of lead. The metal was allowed to attain a temperature sufficient for pouring without forming "cold shuts," and then poured into the mould previously mentioned. This temperature was found to be suitable in all cases when it smoked freely but not excessively. The molten metal ran very free, even better than the brass without lead, probably on account of the increased fusibility. The appearance of this alloy on a surface exposed to the air while melted, such as an ingot-top, was wholly unlike that of brass free from lead; the surface in this case being nearly black and free from the "wrinkles" so prominent in brass free, or nearly free, from lead. The alloy was cold-rolled with considerable difficulty, as a strong tendency to crack was manifested. The bar was planed to a thickness of 0.472 inch and then rolled to 0.352 inch—a reduction of 25 per cent. It cracked on the edges and also "checked" on the surface. Removed the "checks" by scraping and then annealed, pickled, and rolled from 0.352 to 0.247 inch—a reduction of 29 per cent.

"Checked" considerably more during this reduction than before, and the edge-cracks increased in size. Removed the "checks" as before, and then annealed, pickled, and rolled from 0.247 to 0.163 inch—a reduction of 34 per cent.

The "checks" were not as prominent as before, but the edge-cracks were increased in size considerably. Annealed and treated as before, and rolled from 0.163 to 0.111 inch—a reduction of 32 per cent.

The "checks" were now entirely obliterated, but the metal still cracked on the edges to an enormous extent; so much so, in fact, that it was almost impossible to obtain a considerable length of sheet. Annealed, pickled, and rolled from 0.111 to 0.075 inch—a reduction of 32 per cent.

Cracks increased in size. Annealed, pickled, and rolled from 0.075 to .046 inch—a reduction of 38 per cent.

This last reduction was the maximum possible between annealings, as the sheet began to laminate on the end. When this alloy was annealed, drops of lead liquated from the metal, and even became of considerable size before a red heat was reached. Each successive annealing did not seem to lessen the amount. It could be rolled hot, or forged at a cherry-red heat, but cracked badly during the operation. If the heat was other than a cherry-red, it could not be forged at all. The cutting qualities were excellent, the metal was not too hard, the chips were quite short, and a high speed could be utilized; in fact, the properties were all that could be desired in such a composition. The alloy would not be of any service in the arts, however, on account of the extreme difficulty of rolling, the troublesome feature of the lead sweating out when annealed, and the difficulty of obtaining a homogeneous alloy when made in large quantities, due to the tendency of the lead to settle down in the bottom of the crucible, or even in the mould itself. The following results were obtained from the cast and rolled metal:

Bar cast in iron mould (cut from plate).		Annealed Sheet.	
Length,	12 in.		12 in.
Diam.,	0.291 in.	Dimensions,	0.460 by .0465 in.
Sect. area,	0.0665 sq. in.		0.02139 sq. in.
Tensile strength,	1974 lbs.		760 lbs.
Tensile strength per sq. in.,	29,670 lbs.		35,000 lbs.
Elongation in 1 in.,	36 per cent.		20 per cent.
(Fractured section.)		(Fractured section.)	
Elongation in 8 in.,	30 per cent.		16 per cent.
Diam. of fracture,	0.245 in.	Dimensions of frac.,	0.038 by 0.418 in.
Reduction of area,	29 per cent.		25 per cent

Hard-Rolled Sheet.	
Length,	12 in.
Dimensions,	0.0465 by 0.372 in.
Sect. area,	0.0173 sq. in.
Tensile strength,	1100 lbs.
Tensile strength per sq. in.,	63,500 lbs.
Elongation in 1 in.,	2 per cent.
Elongation in 8 in.,	0.3 per cent.
Dimensions of fracture,	0.370 by 0.045 in.
Reduction of area,	4 per cent.

The fracture of this alloy was very short, almost granular,

and the color gray. With a lens, minute globules of lead could be distinguished through the entire mass, more numerous near the bottom of the ingot. In an ingot cast in an open mould were found gray patches near the bottom of the fracture, and yellow ones near the top; the former were proven to be pure lead, and the latter were probably brass containing little or no lead. The color of the polished surface of this alloy was yellow, but of a much lighter tint than the same alloy without the lead. The shrinkage of this composition was greater than that of any of the succeeding alloys.

Experiment B.—Copper 60, *Zinc* 35, *and Lead* 5 *per cent.*

Melted 6 pounds of copper, added 3½ pounds of zinc, and afterwards ½ pound of lead. Poured into the same mould as that used in Experiment A. The melted metal ran very free, and no difference could be distinguished between this alloy and that containing 10 per cent of lead. The appearance of the top of an ingot was quite different from that of the preceding alloy; the color was yellow, with more or less black oxide present; the surface, while not as rough as that of the pure brass, still had more of a resemblance to it than that of the 10 per cent. alloy. The flat plate cast for rolling was planed on both sides until the thickness of 0.456 inch was reached, and the surfaces were perfectly clean. It was then rolled from 0.456 to 0.370 inch—a reduction of 18 per cent.

Began to "check" on the surface, but no edge-cracks appeared. Annealed, pickled, and rolled from 0.370 to 0.290 inch, a reduction of 22 per cent.

More surface "checks" appeared, and also small cracks on the edges. Removed the "checks" by scraping and then annealed, pickled, and rolled from 0.290 to 0.197 inch, a reduction of 32 per cent.

Cracked more on the edges; and the "checks" appeared again. Removed the latter and then annealed, pickled, and rolled from 0.197 to 0.118 inch, a reduction of 40 per cent.

Cracked considerably on the edges, but no "checks" appeared. Annealed, pickled, and rolled from 0.118 to 0.046 inch, a reduction of 61 per cent.

No checks appeared, but the sheet cracked very badly on the edges, and also began to laminate on the end, proving that the

reduction had reached the maximum between annealings. When this alloy was annealed, no drops of lead appeared, such as liquated from the alloy containing 10 per cent. of lead. This composition rolled hot and forged like the previous alloy, that is, only at a cherry-red, and then with difficulty. No difference was noticed in cutting-qualities between this alloy and the previous one; the chips being quite short and the material an ideal one in such respects. The difficulty found in rolling this alloy would hinder its use in the arts as a rolled or drawn material. The lead, however, did not manifest any tendency to liquate in the small bars cast; but it might do so in a larger mass. The following are the results obtained on rolled and cast material:

Cast in iron mould (cut from plate).		Annealed sheet.	
Length,	12 in.		12 in.
Diam.,	0.295 in.	Dimensions,	0.0465 by 0.477 in.
Sect. area,	.0683 sq. in.		0.02218 sq. in.
Tensile strength,	2260 lbs.		934 lbs.
Tensile strength per sq. in.,	33,080 lbs.		42,100 lbs.
Elongation in 1 inch,	28 per cent.		26 per cent.
Elongation in 8 inches,	27 per cent.		22 per cent.
Diam. of fracture,	0.253 in.	Dimensions of frac.,	0.035 by 0.425 in.
Reduction of area,	26 per cent.		33 per cent.

Hard-Rolled Sheet.

Length,	12 in.
Dimensions,	0.046 by 0.576 in.
Sectional area,	0.0264 sq. in.
Tensile strength,	1622 lbs.
Tensile strength per sq. in.,	61,400 lbs.
Elongation in 1 in.,	1 per cent.
Elongation in 8 in.,	Inappreciable.
Reduction of area,	None.

The fracture was very short-grained, similar to that of the preceding alloy. The color of the polished metal was yellow, but of a slightly lighter tint than the 10 per cent. alloy. Minute globules of lead could be discerned in the fracture if examined with a lens, but were not so pronounced or numerous as those of the previous composition. The lead in the alloy seemed to be more thoroughly combined, and no tendency to liquate was noticed in the cast bars. The shrinkage was nominal; and the general appearance was so nearly like that of ordinary brass that the difference was hardly noticeable.

Experiment C.—Copper 60, *Zinc* 37.5, *and Lead* 2.5 *per cent.*

Melted 6 pounds of copper under charcoal and then added 3¾ pounds of zinc and ¼ pound of lead. Poured into mould of the same size as before. Some difference could be distinguished in the running quality between this alloy and the others mentioned before. The outside appearance was similar to the pure brass, except that a trace of black was present on the top of the ingot. The plate was planed on both sides as before and rolled from 0.510 to 0.406 inch, a reduction of 20 per cent.

Cracked considerably on the edges and the surface "checked" a little. Annealed, pickled, and rolled from 0.406 to .330 inch, a reduction of 17 per cent.

Cracked slightly on the edge but did not "check." Annealed, pickled, and rolled from 0.330 to 0.217 inch, a reduction of 34 per cent.

The edge-cracks increased slightly in size. Annealed, pickled, and rolled from 0.217 to 0.137 inch, a reduction of 36 per cent.

Cracks increased considerably in size. Annealed and rolled from .137 to .047 inch, a reduction of 65 per cent.

This last reduction was the maximum, as the sheet began to laminate on the ends. No lead liquated from this alloy when annealed, and it could only be hot-rolled or forged with difficulty, the tendency to crack being nearly as great as that of the preceding compositions. The cutting-qualities were excellent, but not quite as satisfactory as those of the alloys containing more lead. While this composition showed a tendency to crack in cold-rolling not manifested in the alloy free from lead, less trouble would be experienced with it than with the other lead alloys. The tests made on the sheet- and cast-metal gave the following results:

Cast in iron mould (cut from plate).		Annealed Sheet.	
Length,	12 in.		12 in.
Diam.,	0.291 in.	Dimensions,	0.457 by 0.0475 in.
Sect. area,	0.0665 sq. in.		0.0217 sq. in.
Tensile strength,	2590 lbs.		1100 lbs.
Tensile strength per sq. in.,	38,940 lbs.		50,700 lbs.
Elongation in 1 in.,	28 per cent.		27 per cent.
Elongation in 8 in.,	27 per cent.		23 per cent.
Diam. of fracture,	0.247 in.	Dimensions of frac.,	0.365 by 0.397 in.
Reduction of area,	30 per cent.		33 per cent.

Hard-Rolled Sheet.

Length,	12 in.
Dimensions,	0.458 in. by 0.0465 in.
Sectional area,	0.0213 sq. in.
Tensile strength,	1884 lbs.
Tensile strength per sq. in.,	88,400 lbs.
Elongation in 1 in.,	None.
Elongation in 8 in.,	None.
Reduction of area,	None.

The fracture of this alloy was radically different from that of the previous compositions, showing no trace of liquation, and instead of the absence of crystalline formation characterizing the other alloys, this formation was present, and the general appearance of such a fracture was similar to that of the brass without lead, the only difference being the color, which in this case plainly showed the effect of the lead by having a grayish-yellow tint. No globules could be distinguished in the fracture, even with a lens, but the metal seemed to be nearly homogeneous and the fracture was still quite short. The color of the polished metal was yellow, and could not be distinguished from that of pure brass.

Experiment D.—Copper 60, *Zinc* 38.75, *and Lead* 1.25 *per cent.*

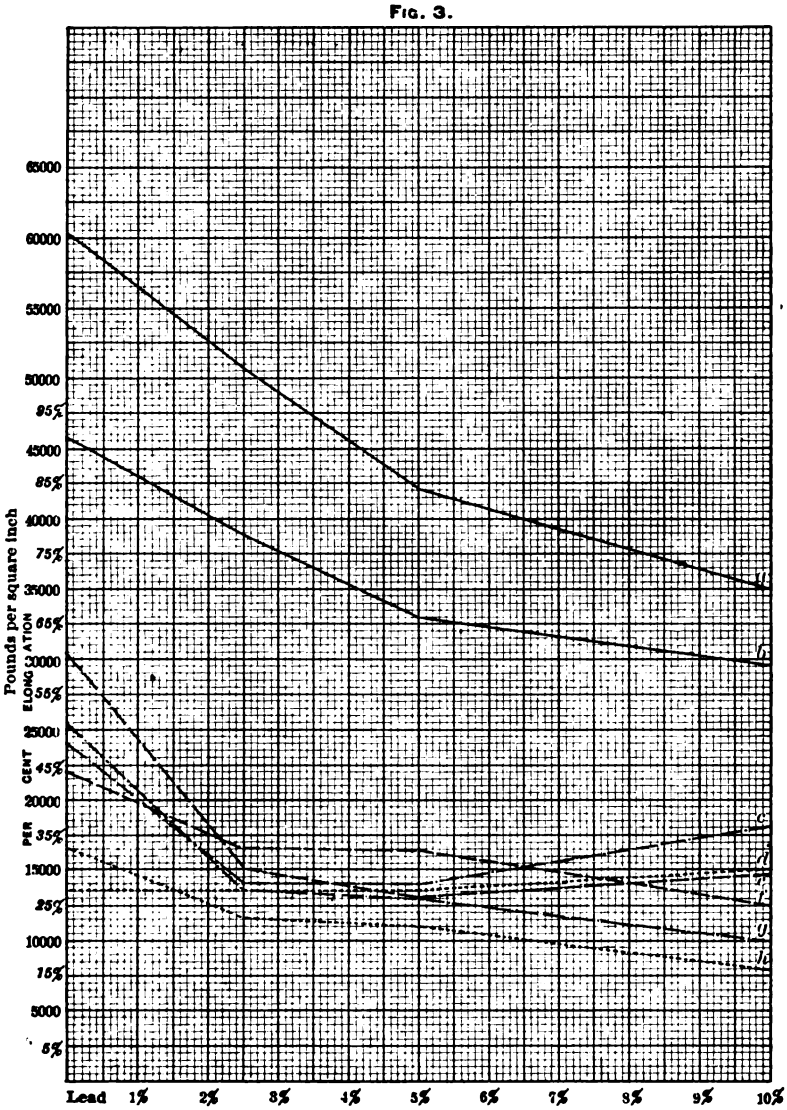
Melted equal parts of the alloy containing 2.5 per cent. of lead, and that containing none (Experiment E.). Poured in the same bar as before, but did not try the rolling process, as the experiment was made for the purpose of investigating the cutting-qualities. In this respect the metal was much inferior to the alloy containing 2.5 per cent. of lead; the chips were short, however, and the cutting-properties considerably superior to those of the pure brass; but the metal was inclined to toughness, and the chips were not as short as necessary for use as a free-cutting alloy.

Experiment E.—Copper 60 *and Zinc* 40 *per cent.*

Melted 6 pounds of copper under charcoal and then added 4 pounds of zinc and poured into a mould of the same dimensions as before. The metal did not run quite as free as the alloys containing considerable lead, but was not a sluggish material, however, no trouble being experienced in running it

into the plate for rolling. Planed both sides of the plate until the thickness was 0.404 inch and then rolled to 0.102 inch, a reduction of 75 per cent. Cracked slightly on the edges, but did

FIG. 3.



Alloys of Copper, Zinc and Lead, Containing 60 per cent. of Copper.

a, Tensile strength of annealed sheet.
b, " " cast metal.
c, Elongation in 1 in. of cast metal.
d, " " 8 in. " "
e, Reduction of area in cast metal.
f, " " annealed sheet.
g, Elongation in 1 in. of " "
h, " " 8 in. " " "

not "check." Annealed, pickled, and rolled from 0.102 to 0.008 inch, a reduction of 92 per cent. Cracks increased slightly in size, but no "checks" appeared. For making the succeeding test, a piece was rolled to 0.043, in order to have the same thickness used in the previous tests, and thus preserve the uniformity.

Cast in iron mould (cut from bar).		Annealed sheet.
Length,	12 in.	12 in.
Diam.,	0.293 in.	Dimensions, 0.483 by 0.043 in.
Sect. area,	0.0674 sq. in.	0.0207 sq. in.
Tensile strength,	3080 lbs.	1252 lbs.
Tensile strength per sq. in.	45,700 lbs.	60,400 lbs.
Elongation in 1 inch,	48 per cent.	51 per cent.
Elongation in 8 inches,	27 per cent.	33 per cent.
Diam. of fracture,	0.181 in.	Dimensions of fracture, 0.373 by 0.031 in.
Reduction of area,	61 per cent.	44 per cent.

Hard-Rolled Sheet.	
Length,	12 in.
Dimensions,	0.0495 by 0.385 in.
Sect. area,	0.019 sq. in.
Tensile strength,	2040 lbs.
Tensile strength per sq. in.	107,000 lbs.
Elongation in 1 inch,	1 per cent.
Elongation in 8 inches,	Inappreciable.
Dimensions of fracture,	0.046 by 0.359 in.
Reduction of area,	13 per cent.

This alloy, known as "Muntz, Yellow or Sheathing Metal,"* exhibited characteristics wholly unlike the alloys containing lead. The fracture was crystalline, the color yellow, and the alloy perfectly homogeneous. The chips were long and tenacious, necessitating a slow speed in cutting. The metal would roll hot or forge at a cherry-red heat, but cracked if the heat was above or below this point. The classification in Table I. will serve to compare the various results obtained. In Fig. 3

*So called after the inventor, G. F. Muntz (see English patent No. 6325 of 1832), and now produced by brass manufacturers, on account of its property of being malleable while hot. In practice it is preferable to add to 100 pounds about 1 pound more of spelter than given above, on account of the danger of volatilization, bringing the alloy too low to hot-roll well. If the alloy contains much more than 60 per cent. of copper it is more liable to crack when rolled hot, but if the content is less the difficulty is avoided, as alloys containing between 50 and 60 per cent. of copper roll hot equally well. On the other hand, if the content of copper (actual) is much less than 60 per cent. the alloy becomes too hard to roll cold satisfactorily, but the proportions given above yield an alloy suitable for this purpose.

TABLE I.—*Summary of Tests of Leaded Brass, Etc.*

Experiment.	E.			C.			B.			A.		
Copper, per cent...	60.0			60.0			60.0			60.0		
Zinc, " ...	40.0			37.5			35.0			30.0		
Lead, " ...	None.			2.5			5.0			10.0		
	Casting.	Annealed Sheet.	Hard-rolled Sheet.	Casting.	Annealed Sheet.	Hard-rolled Sheet.	Casting.	Annealed Sheet.	Hard-rolled Sheet.	Casting.	Annealed Sheet.	Hard-rolled Sheet.
Tensile strength per square inch..	45,700	60,400	107,000	38,940	50,700	88,400	33,080	42,100	61,400	29,670	35,000	63,500
Elongation in 1 inch.....	48 per cent.	51 per cent.	1 per cent.	28 per cent.	27 per cent.	None.	28 per cent.	26 per cent.	1 per cent.	36 per cent.	20 per cent.	2 per cent.
Elongation in 8 inches.....	27 "	33 "	None.	27 "	23 "	"	27 "	22 "	None.	30 "	16 "	0.3 "
Reduction of area..	61 "	44 "	13 per cent.	30 "	33 "	"	26 "	33 "	"	29 "	25 "	4 "
Possible reduction in rolling.....	92 per cent.			65 per cent.			61 per cent.			38 per cent.		

the results have been arranged graphically. From these results it is difficult to discover the reason for the properties produced by the presence of lead in brass. The tensile strength is lowered, but many metals of lower tensile strength than even 30,000 pounds per square inch (copper or tin) are extremely difficult to work. The elongation and reduction of area are also lowered, but many alloys are known of low elongation, whose cutting-qualities are inferior (*e.g.*, genuine Babbitt metal or zinc). Taking the average results obtained in the reduction by cold-rolling, it will be readily seen that the malleability is inversely proportional to the contents of lead. From the fact that in the alloy containing 10 per cent. of lead this element liquated during annealing, and the well-known fact that lead does not produce homogeneous alloys with either copper or zinc, it is probable that this metal forms, with brass, not a homogeneous alloy, but a mechanical mixture. The lead thus prevents the particles of brass alloy from uniting thoroughly and so produces a mass consisting of two separate substances between which there is little or no affinity, both, however, having enough ductility to roll or draw. On the other hand, when the mass is cut, the slight affinity existing allows some of the particles to be easily torn apart. The lead also, undoubtedly, acts as a lubricant, and thus prevents the tool from clogging.

Many authors recommend a slight addition of tin as beneficial in a free-turning alloy, but its use is detrimental rather than otherwise; tin, as is well known, even in small amounts, hardens and increases the tensile strength of brass, qualities not sought for in a free-turning alloy.

Thurston* gives the following alloys as best suited for turned work:

	I. Per cent.	II. Per cent.	III. Per cent.	IV. Per cent.
Copper, ...	61.6	66.5	74.5	79.5
Zinc, ...	35.3	33.0	25.0	20.0
Lead, ...	2.5	none.	none	none
Tin, ...	0.5	0.5	0.5	0.5

In reference to these alloys it may be well to say that brass as low as the two latter compositions should not be used for the base of a free-turning alloy, as the natural softness is not suit-

* *Materials of Engineering*, Part iii., *Brasses and Bronzes*, p. 224.

able. In regard to the three alloys containing no lead it would be well to bear in mind that trouble will follow if such compositions are used for this purpose. An alloy, such as given above, containing only 20 per cent. of zinc and no lead is extremely tough and the chips are quite long; its employment in the arts for a free-cutting material would be ruinous, both to the tools and to the company making the metal.

Chaudet* gives an analysis of a brass intended for lathe-work or filing as follows:

	Per cent.
Copper,	65.80
Zinc,	31.80
Lead,	2.15
Tin,	.25

He says that the presence of tin was probably accidental.

Guettier† mentions an alloy suitable for turning as composed of

	Per cent.
Copper,	61.6
Zinc,	35.3
Lead,	2.5
Tin,	.5

In view of the fact that these authors give tin as an ingredient of brass intended for turning, it may be well to say that no brass intended for this purpose is made at the present time containing tin, and, as previously remarked, its use is to be avoided.

Locke‡ mentions a brass alloy suitable for turning, consisting of

	Per cent.
Copper,	65
Zinc,	33
Lead,	2

Krupp§ gives two formulæ for brass of a quality which he calls "short." He does not say that it is adapted for any particular purpose, but he undoubtedly means that it is suitable for turning. The proportions are:

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* *Manuel de l'Essayeur*, p. 370.

† *Guide Pratique des Alliages Métalliques*, p. 308.

‡ *Workshop Receipts*, Third Series, p. 16.

§ *Die Legirungen*, p. 79.

	I. Per cent.	II. Per cent.
Copper,	63.66	65.5
Zinc,	33.82	32.4
Lead,	2.52	2.1

Bischoff* states that brass intended for fine wire should be free from tin and lead. He analyzed a rod which he states had the quality of "shortness," and obtained the following results:

	Per cent.
Copper,	65.5
Zinc,	32.4
Lead,	2.1

Percy† says that it is usual to introduce a small amount of lead into brass intended for turning, in order that the chips may leave the tool easily. He mentions that it is customary to add three ounces of lead—about 2 per cent.— to 10 lbs. of alloy, and that the addition should be made after the crucible has been removed from the fire, and then afterward well stirred.‡

Kerl§ says that the presence of from 0.10 to 0.30 per cent. of lead in copper renders it both cold- and red-short, and that if containing even 0.10 per cent. of lead it is no longer fit for making fine brass sheet or wire. On the other hand, he says that a small amount of lead seems to increase its facility for rolling and is therefore added to copper of good quality intended for rolling. These two statements contradict one another; and while the first is entirely wrong the second is misleading. Lead is not added to copper at the present time, as its use has been found to be injurious, yet the presence of as small an amount as 0.10 per cent. in copper does not unfit it for use in making brass, for an analysis of cartridge-brass sheet, which was being used by a cartridge-manufacturer with excellent results, gave:

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* *Kupfer*, p. 161.

† *Metallurgy—Copper, Zinc and Brass*, p. 621.

‡ This method of procedure is not followed at the present time, nor is it at all necessary. The lead is added to the mixture in the crucible, after the zinc has been introduced, and while the crucible is still in the fire. The operation is much facilitated by this method, and the alloy produced is perfectly satisfactory, provided the mixture is thoroughly stirred immediately before pouring.

§ *Treatise on Metallurgy*, Crookes and Röhrig's translation, vol. ii. Copper and Iron, p. 81.

	Per cent.
Copper,	66.06
Zinc,	33.55
Lead,	.08
Iron,	.07

If copper containing 0.10 per cent. of lead had been used in making this alloy the composition produced would only have contained 0.06 per cent. of lead. In fact, the content was 0.08 per cent. and yet the alloy was perfectly suitable for cartridge-metal, a material receiving enormous punishment in the process of manufacture. Brass should not contain much over this amount, however, as the liability of cracking in drawing is thereby increased. As no trouble is experienced in making brass containing less than 0.10 per cent. of lead, specifications for high-grade metal should limit the contents to this amount.

An analysis of a rod $\frac{1}{8}$-inch in diameter, and found to be excellent for turning, gave the following results. The metal was manufactured by the maker having the best reputation for this line of work:

	Per cent.
Copper,	62.88
Zinc,	35.03
Lead,	2.09

An analysis of clock-brass made by another concern produced the following results:

	Per cent.
Copper,	61.36
Zinc,	36.22
Lead,	2.44
Iron,	.10

A test of the annealed sheet gave results as follows:

Length,	12 in.
Dimensions,	0.842 in. by 0.026 in.
Sectional area,	0.0219 sq. in.
Tensile strength,	1100 lbs.
Tensile strength per sq. in.,	50,000 lbs.
Elongation in 1 in.,	44 per cent.
Elongation in 8 in.,	38 "
Reduction of area,	50 "

Although clock-brass made after the usual formula will roll hot, if great care is used, so much difficulty is encountered from

cracking during rolling that it has become universal practice to roll the alloy cold, like ordinary brass.

Tests of leaded screw-rods, hard-drawn, gave the following results:

	I.	II.	III.
Length,	12 in.	12 in.	12 in.
Diam.,	0.248 in.	0.233 in.	0.294 by 0.295 in.
Tensile strength,	3,120 lbs.	2,660 lbs.	4,630 lbs.
Tensile strength per sq. in.,	64,500 lbs.	62,400 lbs.	54,000 lbs.
Elong. in 1 in.,	30 per cent.	27 per cent.	46 per cent.
Elong. in 8 in.,	10 per cent.	13 per cent.	26 per cent.
Diam. fract.,	0.160 in.	0.140 in.	0.200 by 0.204 in.
Red. area,	58 per cent.	63 per cent.	53 per cent.

These samples were made by different manufacturers.

The increasing use of automatic cutting-machinery, etc., has demonstrated the fact that the best results are only to be obtained by the employment of an alloy containing lead, and the most successful manufacturers of such material are those who are able to introduce the maximum amount of lead into the mixture and yet be able to roll or draw the product. The formulæ now used by brass-makers for "leaded stock" are almost identical with one another, but the product often differs through faulty manipulation. The proportions are as follows:

	Per cent.			Per cent.
Copper,	60.0		Copper,	60.0
Zinc,	38.5	or	Zinc,	38.0
Lead,	1.5		Lead,	2.0

or, in the vernacular of the brass mill, a two-and-three brass with from 1½ to 2 pounds of lead added to the hundred. The usual method of making this alloy is to weigh out, say 60 pounds of copper, and introduce 40 pounds of spelter, with enough more to compensate for the loss by volatilization (casters usually add about ½ pound in excess, but this is hardly sufficient in the majority of cases; see footnote, p. 496), and then add the lead to the molten mixture. This method, of course, gives slightly less than the above amount.

Some manufacturers prefer to make the base a five-and-eight*

* This phraseology is employed in brass-casting to designate the parts of the various ingredients taken. For an example, a five-and-eight brass is one composed of 61.54 per cent. of copper and 38.46 per cent. of zinc. The method is used to facilitate weighing, as it is difficult to obtain a given weight of copper ingots on

brass, and then add the above percentage of lead, while still others use even a two-and-one brass; but the alloy containing 60 per cent. of copper has always given the best satisfaction.

The precautions necessary for the manufacture of the best quality of screw-rod or clock-brass are as follows:

The content of copper should not be below 60 per cent. or above 70 per cent., and the lead should not vary more than 0.25 per cent. from 2.00 per cent. Spelter should form the remainder.

The alloy should be free from tin, or not more than 0.10 per cent. of tin should be present. Tin is extremely liable to find its way into brass not supposed to contain it, on account of the different grades of brass made in the casting-shop, such as spring-brass, etc., which contain tin, and the scrap from these metals finding its way into other mixtures in small amounts.

As much scrap as possible should be employed, as such material tends to increase the shortness of the chips. While in ordinary brass it is not usual to introduce more than 25 per cent., as much as 50 per cent. of scrap can be added to screw-rod or clock-brass to a good advantage, if the scrap is of good quality.

The alloy for rod should be as hard-drawn as possible, and in the majority of instances the sheet should be hard-rolled.

The tensile strength should not be above 70,000 pounds, or below 50,000 pounds, per square inch, and the elongation not more than 25 per cent., or less than 5 per cent., in 8 inches (not applicable to large rod above 1 inch in diameter).

By following the above specifications satisfactory brass for turning or cutting of any description will be obtained. It may be well to add that although only 1½ pounds of lead are usually added to 100 pounds of alloy, the content of this element often rises to 2 per cent. on account of the presence of lead in the spelter.* The amount of lead in the spelter should be known in order to obtain the right amount of it in the brass.

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account of their uniformity of size. In making brass according to the above method a certain amount of copper is weighed within a few pounds more or less, but whatever the weight is it must be accurate; the proportion of zinc called for in the formula is then weighed with accuracy, an operation easily performed on account of the brittle nature of this metal.

* An analysis of a sample of spelter used for making cheap brass gave 1.34 per cent. of lead.

The method of making leaded brass is as follows: The copper is melted in a plumbago crucible under a good layer of charcoal,* and when hot enough to avoid chilling (no hotter) the spelter is added and then the lead. The mixture is next stirred thoroughly with an iron stirrer, and, if necessary, allowed to get hot enough before pouring to run without forming cold shuts, otherwise a "spilly" bar will result. The determination of this temperature is the most difficult problem of brass-casting, and is the most important point in the operation, for if the metal is allowed to get too hot it will either crack in rolling or become "spuey," that is, full of blow-holes; or often both these faults will be present. On the other hand, if poured too cold the divided metal falling to the bottom of the bar will not unite and the metal will become "spilly," that is, covered with splinters or scales when the metal is rolled or drawn, due to the ununited metal being extended. The determination of this temperature is made by the eye, and no other method for obtaining it, which is satisfactory, is known at the present time. Other points, such as the size of the stream in pouring, oil on the mould, temperature of the mould, etc., all have influence on the finished metal; but their discussion is out of place here.

For rod above medium size, say ¼-inch in diameter, it is customary to cast what is called a "bolt" for making the rod. This is a bar from 1½ to 2 inches in diameter, depending upon the size of the finished product, and about 6 feet long. This bolt is drawn on draw-benches or rolled through grooved rolls until of the required size. For small rod or wire it is usual to cast a "wire-bar," that is, a flat plate from 3 to 4 inches wide, 1 to 1¼ inches thick and about 6 feet long. These bars are rolled cold until the thickness has reached ⅝ or ½ inch, depending upon the capacity of the slitters, and then slit into square strips, which are afterward edged and drawn on the regular wire-block. Sheet is made from a plate in the usual way.

Brass containing lead tarnishes more readily than the pure material; it is also claimed that such an alloy is better suited

* Many brass makers use charcoal of the fineness of pea-coal as a covering for the metal. This quality, while unsuitable for the majority of uses to which such material is put, is excellent for a covering for melted metal, on account of its greater protection to the molten material from the gases and air, than that given by larger lumps of charcoal.

for ships' sheathing than brass containing no lead. This fact has been the subject of a patent issued to G. F. Muntz,* an extract from which is as follows:

"My invention has for its object to improve the manufacture of my sheathing metal, for which Letters Patent† were issued, which consisted of employing an alloy of copper and zinc in such relation to each other that, whilst the copper was to a considerable extent preserved, there was a sufficient oxidation to keep the bottom of a ship clean, and in practice it was found by me to require as nearly as might be 60 parts of copper to 40 parts of zinc for that purpose, and such proportions of copper to the zinc could not be reduced, even to a very small extent, without rendering the alloy liable to have the zinc separately acted upon, which was highly prejudicial. Now the present invention consists of using another suitable metal or metals so as to allow the mixture to contain a less proportion of copper than 60 parts and at the same time to have a sufficient degree of oxidation and prevent separate action on the zinc, thereby to keep the bottom of a ship or vessel clean. . . . . This alloy consists of 56 parts of copper, 40¾ parts of zinc and 3¼ parts of lead. . . . . I would remark that the lead in the above compound or alloy acts a very important part, as an alloy consisting of 56 parts of copper and 40¾ parts of zinc would not without the lead produce a metal which would oxidize sufficiently to keep the bottom of the ship clean. The alloy being made and cast into ingots, it is to be rolled into sheets, preferably at a red heat, and the sheets produced are to be annealed. . . . . I do not claim the combination of lead or other metals with copper and zinc for sheathing, except when made after the formula given in this invention."

While the composition of the alloys were different from the formula proposed by Muntz, the following experiments may have bearing upon this point. Pieces of sheet about 0.03 inch in thickness and 2 inches by 4 inches in size, of the following alloys:

	A. Per cent.	B. ‡ Per cent.
Copper,	63.60	61.36
Zinc,	36.13	36.22
Lead,	.21	2.44
Iron,		.10

were immersed in sea-water for 200 days and then cleaned and weighed. The corrosion seemed uniform on each and they were not pitted. The relative corrosion was as follows:

A.—Loss in grammes per sq in. of surface,	0.0058	or 1 to 1.05
B. " " " "	0.0055	

* English patent, No. 11,410, of 1846, to G. F. Muntz.

† English patent, No. 6325, of 1832, issued to G. F. Muntz.

‡ See p. 501. These two samples were identical.

The corrosion was therefore nearly the same and the effect of the greater amount of lead not apparent.

The fractures of the alloys containing lead, broken while red-hot, are interesting. The alloy without lead gives a fracture of a crystalline nature, and is not tarnished perceptibly; in many cases the color seemed as if the metal had been dipped in acid. The fracture of the alloy containing 2.5 per cent. of lead was similar to that of the pure brass, but the color was darker and the appearance not as crystalline. That of the alloys containing 5 and 10 per cent. of lead was very different from the others; the appearance was almost amorphous, the color black, and the general look more of cast-iron broken hot than that of brass.

Percy* gives an analysis of a Chinese brass sheet about $\frac{1}{16}$ inch in thickness.

	Per cent.
Copper,	56.59
Zinc,	38.27
Lead,	3.30
Tin,	1.08
Iron,	1.47

The lead in this alloy was probably accidental.

As previously observed, leaded brass, in order to give satisfactory results, should not contain less than 2 per cent. of lead. The following analysis was made of rod used on a screw-machine, which did not give quite as good results as previous lots:

	Per cent.
Copper,	65.36
Zinc,	33.18
Lead,	1.46
Iron,	not determined.

It must not be supposed, however, that brass containing less than 2 per cent. of lead will not work, for such is not the case; in some cases an alloy containing as low as 1 per cent. could be used, but in such a case one containing 2 per cent. would give better results. In many cases it is impossible to get satisfactory results unless the content is at least 2 per cent. It is better, therefore, to always work with a single formula. The following analysis was made on the above material after being

* *Percy's Metallurgy—Copper, Zinc and Brass*—p. 618.

melted. The chips from the rod were the material which were melted, and no large pieces were used, the entire mass being chips of ordinary fineness. The crucible was kept full of chips continually, and the temperature was not allowed to become excessive; the mass smoked freely when poured, but not for any length of time. The analysis is as follows:

	Per cent.
Copper,	67.99
Zinc,	30.15
Lead,	1.76
Iron,	.16

As the zinc is the most volatile element, the fact is apparent that the lead increases in amount, and therefore leaded brass which contains less than the required amount of lead may, by melting, increase it so as to be serviceable.

The incorrect amount of lead may often be introduced into brass by not stirring thoroughly after the lead has been added to the crucible. The lead when first introduced into the mixture settles to the bottom, and when poured into the bar the bottom portion is the last to be poured, and thus the top of the bar will contain more lead than the bottom. The whole bar will roll or draw, and therefore the defect will not be noticeable until the consumer begins to have trouble in cutting. A case of this description recently came to the author's notice, where an analysis of the top of the bar gave 1.87 and the bottom 1.06 per cent. of lead. The amount of lead added was about 1.5 per cent.

In order to discover whether lead has a tendency to combine with brass in any particular proportion a rod about $\frac{5}{8}$-inch in diameter and 6 inches long was turned from the alloy used in Experiment B (containing 5 per cent. of lead) and inserted in a hole whose diameter was slightly larger than that of the rod, drilled in a 2-inch cylinder of crucible-material. The mouth of the hole was loosely stopped with sand, and the whole arrangement set in a No. 25 crucible and heated until the fumes of zinc oxide (spelter fumes) were given off freely. The crucible was then filled with hot sand, completely covering the metal, and giving a depth of several inches all around the cylinder. The crucible was covered, and allowed to cool very gradually by leaving in the fire until cold. The bar of metal was then

removed and examined. No lead could be seen either on the bottom or top, but the bar had every appearance of a homogeneous mass. It was expected that the lead would partially settle down to the bottom, and that the top would contain the true proportion of the lead in the alloy. Such was not the case, as the following analyses will show:

	Bar before heating.	After heating.	
		Top.	Bottom.
	Per cent.	Per cent.	Per cent.
Copper, ...	61.03	61.09	60.94
Zinc, ...	33.50	32.46	33.40
Lead, ...	5.38	6.45	5.66

The lead had not floated to the top as would at first seem, but the loss of spelter was greater in this portion than at the bottom. Whatever the cause of this action may be, it is evident that lead forms a homogeneous mixture with the brass if not a definite alloy.

No brass is made at the present time which does not contain more or less lead, ranging from the small amount contained in cartridge- or spinning-brass to that of clock or screw-rod material. The reason for this is, that it is impossible to obtain zinc free from lead. It may be said that one is as sure to find lead in brass as phosphorus in steel.

The Technology of Cement Plaster.

BY PAUL WILKINSON, ST. JOSEPH, MO.

(Lake Superior Meeting, July, 1897.)

History.

From the earliest times, the principal component of wall-plaster has been ordinary lime. Plaster-of-Paris has also been known from early times, but never used to any extent in the actual base-work of plastering-materials; it is generally used for finishing over the brown coat and for ornamental work. As an illustration of the antiquity of the use of gypsum-plaster for mortar, I find that the analysis of the exterior covering of the pyramid of Cheops, as made by Dr. Wallace, is almost identical

with that of our best cement-plaster. The analysis is given in the *American Encyclopædia* as follows:

Hydrated calcium sulphate,	82.89
Carbonate of lime,	9.80
Magnesia,	0.79
Silicic acid,	4.30
Alumina,	3.00
Oxide of iron,	0.21
Total,	100.99

For the last quarter-century, however, plaster-of-Paris has been the chief constituent of numerous patent plasters, or "hard plasters," as they are sometimes called. The formulæ of these all vary but slightly. In addition to plaster-of-Paris, we find in the records of the Patent Office the following substances used to perfect the working-qualities of these materials, or to give them some peculiarity for the purpose of advertising, viz.: slaked lime, sawdust, asbestos, and various qualities of clay, together with some form of retarder to restrain the setting-action of the plaster-of-Paris for such a time as will give the mechanic opportunity to apply the mortar nicely.

As early as 1871 gypsum-rock had been found and manufactured in Kansas. Later, a workman from one of these plaster-of-Paris mills, which were then all located at Blue Rapids, in the eastern part of the State, happened to be stopping at the farm-house of Mr. Jonathan Tinkler, 1¼ miles northeast of the present town of Gypsum City, Kansas, and his attention was called to some plaster which Mr. Tinkler had made for plastering the cellar of his house, by calcining in an ordinary 30-gallon kettle some of the so-called dirt from one of his fields. That plaster is still there and in good condition. Mr. Tinkler discovered this earthy deposit while running a "fire-guard" around the field; he did not manufacture the material, however, until 1875. The above-mentioned workman told Mr. Tinkler of the importance of his deposit; and in 1889 the latter interested some local capitalists, a small company was formed, and a discarded plaster-of-Paris mill from a nearby town was purchased and moved to Gypsum City, where it was set up for the treatment of this earth, the crude material being hauled in wagons from the farm to the mill in town. This original company in this field was known as the Saline County

Plaster Co.; the deposit being located in Saline county, Kansas.

In casting about for a proper name for this material, which, by the way, when manufactured, resembles a Portland cement, the manufacturers finally decided upon the name "Cement Plaster." I believe their idea was chiefly to obtain the advantage of whichever freight-rate, that on cement, or that on plaster, should be the lower, as well as to provide for the possibility that the material should prove a successful cement. This name, Cement Plaster, was afterwards applied by the plaster-of-Paris companies to their wall-plaster products, but rightfully belongs to only those wall-plasters which are manufactured from this earthy material.

During a recent conversation, my friend, Dr. E. H. S. Bailey, Professor of Chemistry in the State University at Lawrence, Kansas, suggested that we call this material "gypsite," so that it might have a distinctive name. I think this suggestion a good one; have endeavored to introduce this name in the industry, to denote this gypsum-dirt; and will so use it throughout this paper.

In 1890, the Saline County Plaster Co. sold out to a company organized by larger capitalists, and called the Acme Cement Plaster Co., which has been in successful operation ever since. After the Acme Co. had developed a successful industry from this deposit, others began to investigate, and other companies were formed. It was thought for a time that the Gypsum City deposit was the only one in existence, and practically inexhaustible. It covered an area of 12 acres with an average thickness of 8 feet, and more than 100,000 tons were manufactured from it before it was abandoned. The original members of the Saline County Plaster Co., after they had sold out to the Acme Co., prospected in the "Pan-Handle" of Texas, discovered a large and valuable deposit there, and formed the Lone Star Plaster Co., operating at Quanah, Texas. This was absorbed by the Acme Co. a few years later. The deposit, covering in all about 6000 acres, is certainly one of the most remarkable ever discovered, by reason of its vast extent and the character of its material, and the Acme Cement Plaster manufactured here is shipped to all parts of the United States, and even to Canada. An experimental shipment has just been made to Australia and New Zealand.

The field is now covered by several companies, manufacturing as follows: The Salina Cement Plaster Co., at Dillon, Kan., 14 miles east of Gypsum City; the Dillon Cement Plaster Co., 3½ miles southwest of Dillon. The Acme Co. has another mill 3 miles beyond the last-mentioned. The Salina Co. also has a second mill, at Longford, in Clay county, Kan., 35 miles northwest of the Dillon mill. The Acme Co. has its principal mill 5 miles northwest of Quanah, Texas, which town is 192 miles northwest of Fort Worth. In this vicinity there are a few temporary mills experimenting, and there are also a few mills in the Indian Territory, the only one of any importance being that of the Royal Cement Plaster Co., at Marlow. There are at present eleven mills in Kansas engaged in the manufacture of plaster; four use the gypsite and seven the gypsum-rock.

Indications all point to the final location of the whole of this industry in the "Pan-Handle" of Texas, for the reason that the Kansas and Indian Territory beds are comparatively limited, whereas the deposits seem to exist in inexhaustible quantities in the "Pan-Handle."

General Conditions.

This paper is intended to deal exclusively with the technology of the cement-plasters. As I understand, the geology of the subject has been and will be treated by others. Hence I will not discuss the character of these deposits, beyond observing that there is a great diversity of opinion as to their origin. The most plausible theory seems to be that of their precipitation from springs, aided by wash from adjacent territory.

The composition of the crude material, as well as its manner of deposit and its proximity to railroads, are all points to be considered by the manufacturer. *Per se*, these industries do not warrant at the present time the construction of independent lines of railroad. Many of the deposits lie in low, swampy ground, and gypsum-springs usually occur in them. In most cases there is a ledge of gypsum-rock at the bottom of the deposit, but sometimes even lower, and sometimes not immediately under, but on the same level, at some distance from the gypsite-deposit. The deposit of the Acme Co. in Texas has

an average depth of 30 feet, and immediately under this is a heavy bed of excellent gypsum-rock, which is used by the company for the manufacture of plaster-of-Paris. Many deposits have been found which are entirely worthless for one or both of two reasons: because of the percentage of sand and clay, where the above-mentioned side-wash is excessive; or because the material occurs in too small pockets. This material being worked by plow and scraper, too much labor is required to operate small pockets properly. In these sections the water of all the streams and wells is highly impregnated with magnesium sulphate, which does not seem to affect animals, but has the usual effect upon man. In most instances, however, the workmen become habituated to the use of this extremely mineral water.

Technology.

In considering the technology of gypsite it is necessary to consider the composition of the material in comparison with that of gypsum-rock, in order to understand the difference in the manufacturing processes of plaster-of-Paris and cement-plaster. It is the foreign ingredients in the gypsite which cause the difference. The analysis of a sample of first-class gypsite is as follows:

	Crude.	Manufactured.
Insoluble residue,	.91	2.53
Iron and aluminum oxides,	.21	.45
Calcium sulphate,	69.92	78.81
Calcium carbonate,	9.05	11.22
Magnesium carbonate,	.65	.74
Water,	18.85	5.70
Total,	99.59	99.45

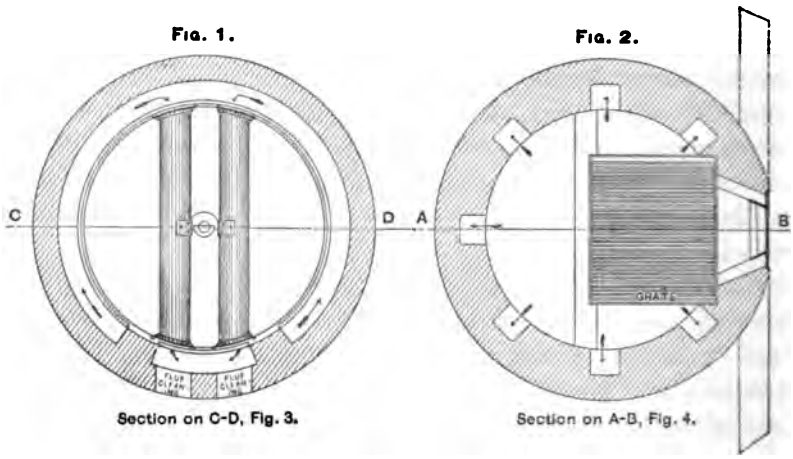
Gypsum-rock, as is well known, occurs in ledges; is mined and transported to the factory; and is there reduced by nippers and Gates crushers to the size of small marbles. It is then run through buhrs and finely ground, and conveyed by elevators to bins over the calcining-kettles. From this stage, the process is practically identical with the manufacture of gypsite into cement-plaster. Gypsite occurs in pockets of greater or less extent on the surface of the earth, with from a few inches to 2 or 3 feet of surface-loam on top. It resembles marl in appearance, and is in a fine state of subdivision. It is usually more or less moist, on account of the fact that the pockets

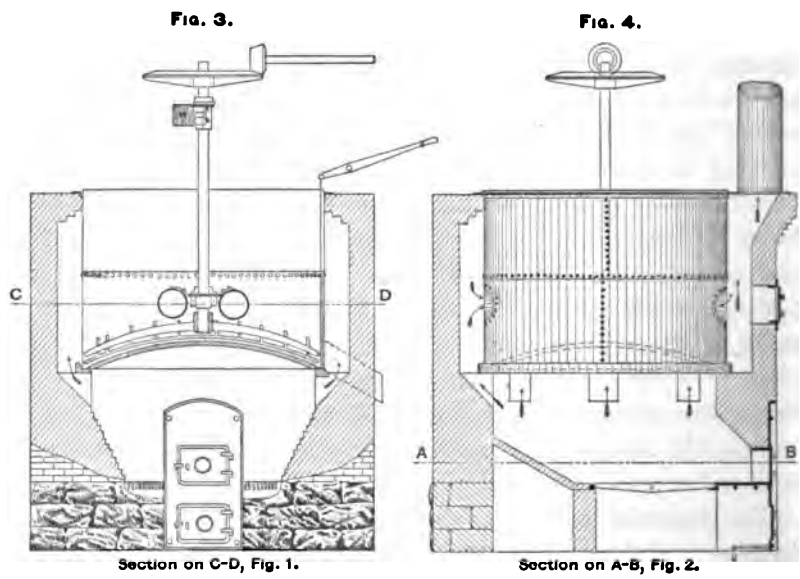
occur in marshy meadow-lands along the sides of a stream. In mining it, the surface-covering is stripped off and scraped to some distance outside the deposit. It is most advantageously manufactured during the hot season of the year, as at that time manufacturers obtain the assistance of nature in drying out the physically-combined moisture. It is a peculiarity of gypsite that it absorbs and holds water with great avidity, and freezes readily, so that in securing stock for winter purposes it is necessary to have large storage-sheds and fill them during the drier months of the year; hence the scene of greatest activity is during the months of July, August and September, when the manufacturers endeavor to fill their sheds with dry material, and at the same time are being "rushed" by the large demands for their product, as the building-season is then at its height. The material is frequently piled in various portions of the bed to mix the various strata thoroughly, as well as to permit the moisture to drain out. I have seen these heaps frozen to the depth of 2 or 3 feet, in weather when ordinary soil would not be frozen at all.

Manufacturing-plants, when possible, are located on the gypsite-bed itself, or adjacent thereto, so that the expense of the transportation of the raw material may be reduced to a minimum. No matter what the distance from the railroad, it is cheaper to transport coal from the railroad to the mill as a back-haul and to haul the manufactured material, than it would be to haul the crude, wet material from the bed to the railroad and manufacture there. This has been illustrated in the experience of the Acme Co., which hauled its crude material 1¼ miles to its mill at Gypsum City, whereas its other Kansas plant is about 7 miles from the railroad, and loads of two or three tons are hauled on broad-tired wagons to the shipping station, the hauling-contractor making a very favorable rate on coal and supplies for a back-haul.

The material is loosened in the deposit by means of disk-harrows and then taken into the stock-warehouse by means of wheeled scrapers. There are some irregularities in every deposit; and the principal object sought by the manufacturer is to mix the materials thoroughly, so as to procure a uniform output. After the material has been put in the warehouse it is at the same stage as the plaster-of-Paris, when the latter has reached the bins, as above described.

The principal and really exact process is that of calcination. For this purpose boiler-iron kettles (see Figs. 1 to 4) of $\frac{5}{8}$-inch thickness, 8-feet diameter and 6-feet depth are used. The





SECTIONS OF CALCINING KETTLE.

foundation for a kettle is necessarily built very solid, the fire-space within the foundation being in the shape of an inverted truncated cone 4 feet high. At the top of this a cast-iron

flanged ring is set into the masonry. The kettle-bottom, consisting of a concavo-convex iron casting, a little less than 8 feet in diameter, with convexity placed upward with a rise of 1 foot, and with a thickness of $\frac{3}{4}$ inch at the edges and of 4 inches at the crown, is set within the flange at the top of this ring. The kettle proper is then placed over the kettle-bottom. This kettle has two flues 12 inches in diameter, placed transversely about 8 inches above the crown of the kettle-bottom, and separated externally about 6 inches. After the kettle has been set, brick masonry is erected around it, gradually converging at the top to meet the top-rim of the kettle. The first floor of each mill is usually built around the kettle about a foot from the top, sometimes level with the top, to facilitate the shoveling of dirt directly into the kettle by hand; and the kettle, with the furnace, is in the basement, with storage-room for fuel conveniently arranged in front of the kettle. Ports are made through the side of the base-ring; and the heat from the furnace is deflected by bridges around the surface of the kettle, so that it may cover every part of the kettle, pass through the flues, and finally make exit through a regular stack. For fuel, the best of coal must be used, having a minimum of sulphur, coking freely and giving a long flame. The best coal is procured in the Trinidad district of Colorado. Kettle-bottoms of sheet steel have been tried, but do not serve as well as cast-iron ones. Only the best scrap-iron must be used in these kettle-bottoms, it having been found that ordinary scrap-iron does not last nearly so long as pig. The life of a kettle-bottom is terminated by cracking. The cracks can be caulked with asbestos cement, but the expense of repairing and stoppage soon overcomes the saving. At the top the kettle is covered with a sheet-iron cap with a movable door, through which the material is introduced, usually by a chute, fed by an elevator from the dirt-pit. The usual number of kettles is four to six. They are arranged in line and usually worked in pairs. It is necessary that the material in the kettle be constantly agitated; and for this purpose a line-shaft is run over the kettles and a vertical shaft runs from this to the bottom of each kettle, being supported below by a saddle placed between the flues. At the bottom of the shaft a curved cross is attached, to which are affixed movable teeth with paddles, which are so adjusted

as to throw the material from the periphery to the center. Should the agitation stop, or the teeth become broken, the material settles down on the bottom, and, owing to the intense heat, the bottom is almost instantly melted through. The material when heated is very fluid, runs through like water and quenches the fire. The Acme Co. had in one instance a heavy loss in the death of a fireman who was slashing the fires at the time when a kettle-bottom burned through. The material ran down, was deflected from the fire, and the fireman was instantly enveloped and burned to death. The stoppage of the agitation can usually be detected by the calciner, who stands above and is supposed to watch the process of calcination constantly; but in this instance the calciner had stepped aside for a moment, and it was all over when he returned.

Both water-power and steam-power are used for driving the machinery. When steam is used, the slack from the fuel used under the kettles is employed for the boilers.

Upon the correct calcination, more than anything else, depends the quality of the material. The calcium sulphate in gypsite is hydrated, as it is in gypsum-rock. The object sought is to drive off a portion of this water of crystallization. To attain this end, in the plaster-of-Paris manufacture, the temperature does not exceed 340° F.; but in the manufacture of gypsite, probably on account of the foreign matters, a higher temperature is required, which averages close to 396° F. In starting a kettle, heat is gradually applied, and the crude material is gradually fed in and constantly agitated. This process is slow, and requires some length of time, on account of the vast amount of physically-combined moisture to be evaporated. Material is gradually added until the kettle is full, and during this process the contents of the latter boil in a violent manner closely resembling the boiling of water. The fact that the physically-combined moisture is all evaporated is indicated by the prompt settling of the material; the heat then rises to a higher degree and ebullition again takes place, indicating the driving-off of the water of crystallization. The point at which the process is complete is known by the manner in which the material boils and by its general appearance; and the calciner, at the proper moment, lets off the charge through a small gate at the bottom and in the side of the kettle. Thermometers are

used in plaster-of-Paris manufactories to govern the temperature exactly; but in the gypsite-manufactories the point varies slightly and is usually best known by an experienced calciner. The escaping steam is let off by means of a stack let into the sheet-iron cover of the kettle parallel with the smoke-stack, and this stack contains near its base a separator similar to the steam-separators, for the purpose of retaining the plaster-dust. It has been found by raising the iron cover about 18 inches and putting on proper sides, that it furnishes a chamber above the boiling material and greatly assists the escape of the steam from it.

From the kettle the hot material runs like water into a fireproof pit. The kettles are usually run in couples so that one pit will do for two kettles; and one chute will do for two kettles in filling, as the kettles are run at slightly different periods. Each kettle contains a charge of about five tons of manufactured material, and requires about three hours to calcine properly. After cooling slightly, the manufactured material is elevated into a revolving screen, which separates all small particles and foreign matter, and renders the product uniform. The screenings run from $\frac{1}{2}$ to 1 per cent. only. It is usual to have a series of screw-conveyors and elevators both in front of and behind the screen, so as to mix the material thoroughly. Owing to the temperature of the material, all conveyors, elevators and interior linings must be of metal, and the screen is made of wire-cloth. From the screen, the material is conveyed to the storage-bins, which are usually arranged to hold 100 or 200 tons, and of which there are several, so as to separate, if desired, the runs of different days. The material is usually allowed to fall from a screw at the top of the building, first, that it may spread out and let the different portions mix thoroughly, and, secondly, that it may cool in passing through the air.

Various elevating devices have been experimented with, among them the blower. It was proposed to use this for the purpose of elevating the hot material from the kettle-pit to the bins, as it was hoped by this means to render unnecessary a screen and expensive bucket-elevators, and it would also cool the material. This method was tried at one of the plants, but was soon abandoned, as I infer, on account of the clogging of the material, which, owing to the fineness of the manufactured

article, diffuses through the atmosphere, soon fills up all open parts of machinery, and renders efficient lubrication extremely difficult. The friction in certain parts is heavy, and the bearings must be watched carefully.

Owing to the peculiar characteristics and constituents of different deposits, the setting-time of the manufactured article is apt to vary greatly. The most suitable time for working the material is about one hour, when it is mixed with sand. This gives the plasterer time to apply the mortar properly, and after that time the material commences to harden, so as to render further tempering impossible. Manufacturers whose material does not reach this standard, usually have to "retard" their material. For this purpose there are numberless inventions. Some of these retarders are added to the material just before it leaves the kettle, so as to have the advantage of the thorough mixing which it receives in its course to the bins. Others are applied just before the material goes into the storage-bins; and some, after it has been taken from the storage-bins and just before it is sacked. Some manufacturers add hair or fiber, as desired by the customer. The material is then sacked directly from the mixer.

The final process in the production of this material is the packing. Barrels are seldom used, the principal containers being jute sacks, which contain 100 pounds net. These sacks are made of the best jute, and will stand considerable wear and tear in transportation. They are sold to the consumer, and he is allowed a credit of the same amount as he paid for them, on returning them to the mill. Thus the manufacturer is enabled to keep the burden of the package on the consumers. Paper sacks are also used, as in the hydraulic cement industry. These contain but 75 or 80 pounds of cement-plaster, however, as the material is extremely light and fluffy. These sacks are not desirable, as for some reason, when filled with cement plaster, they break and tear much more easily than with hydraulic cement.

Remarks.

In conclusion I would observe that the fuel-question is the one above all else demanding attention. Owing to the simple arrangement of the kettles but a small amount of the heat is utilized; and by some arrangement for dryers arranged over

the kettles the crude material could be made to absorb and utilize a good amount of this waste. Various plans have been proposed, but none of the manufacturers have as yet felt the necessity of going to the expense of experimenting with any of them. As in all other industries, however, the value of economy will probably be realized at all points, as profits become smaller.

The recent discovery of fuel-oil in the districts in which the plants are located seems to point to the adoption of this fuel in the near future by the cement-plaster industry. Experiments were made by the Acme Co. at one of its mills, some time ago, with oil for fuel; but, owing to the uncertainty of the contract it was able to secure with the producers, the company did not feel warranted in going to the expense of equipping its plant with the proper appliances, and adopting oil as fuel.

Much remains to be done in the way of a careful and thorough study of the exact chemical composition of the gypsite, and the best methods for combining the various portions of the deposits, as the setting-time seems to be greatly influenced by variations in the composition, and this setting-time is the all-important point to be controlled.

The Iron-Ore Supply.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(Lake Superior Meeting, July, 1897.)

FORTY years ago, when the first shipments of iron-ore were made from the Lake Superior region, the supply for the blast-furnaces active at that date was in most cases a local consideration; the majority of the smelting-plants being close to the ore-mines. Of those which depended in whole or in part upon ores brought from distances greater than an economical wagon-haul, few obtained supplies by canal or railroad from points 50 miles removed. At the present time the ore-supply is a local consideration for but a small proportion of American blast-furnaces, and 700 miles or more separate a greater num-

ber from the mines producing the ore in 1897 than was supplied in 1857 with ore carried 50 miles.

The iron-producing district of which Birmingham, Ala., is the business center is the most prominent section now dependent upon local ores. Other blast-furnaces in Alabama, Georgia, Tennessee, Virginia, Pennsylvania, Ohio, Kentucky, Missouri, Wisconsin, New York and New Jersey obtain all or a portion of their quotas of iron-ores from mines located within short distances.

It is not necessary to detail in this paper the wide distribution of ores from the Lake Superior region, which now contributes five-eighths of the annual production of domestic iron-ore. Let it suffice to say that, taking into account all the ore mined in this district, the average transportation by rail and water exceeds 800 miles between mine and furnace, and probably three-fourths of this ore is subjected to three and sometimes four transfers between the time when the loaded cars leave the mine and the time when other cars deliver the ore at the furnaces. That is, a large proportion of the ore mined in the Lake Superior region is loaded on cars which, after covering 15 to 100 miles of railroad-haul, are discharged into the pockets of ore-docks. These pockets hold the ore until vessels are ready for cargoes. When the receiving-docks of the lower lakes are reached, the ore is removed from the vessels into cars, or placed in stock-piles from which cars are loaded subsequently, as the consumers require the ore, and is finally unloaded at the furnaces.

If, in addition, we consider the loading of ore underground in mine-trams and skips, the hoisting of the same to the surface, the dumping of skips into shipping-pockets or stock-piles (in some cases sorting the ores), and the subsequent loading of railroad-cars at the mines, the number of transfers of much of the ore obtained in the Lake Superior region is from five to ten. Where steam-shovels are employed in open cuttings, the handlings may be reduced; but notwithstanding this apparent economy, ores can only be mined and carried for the distances above named by the employment of efficient machinery, appliances and management, when the basis-price for Bessemer ore (yielding when dried at 212° Fahr. 63 per cent. of iron, and phosphorus 0.045 per cent.) at the receiving-docks at the lower lake ports is $2.40 per long ton.

The production of iron-ore in the Lake Superior region in 1896 exceeded that of any former year, the various mines contributing 10,566,359 long tons, and bringing the total output since the first shipments in 1856 to nearly one hundred and ten million long tons.

When the Institute first visited the Lake Superior region in 1880, the Marquette range had in twenty-five years shipped fourteen million tons of ore, of which 1,384,000 tons was credited to that year; and the Menominee range, then in its fourth year of development, had contributed a total of less than one million long tons, of which more than half was forwarded in 1880.

Four years subsequently to this visit, the Gogebic range in Michigan and Wisconsin and the Vermilion range in Minnesota were added to the producing territory, and three years thereafter, when the Institute met at Duluth (July, 1887), the various ranges had made the following record:

Year of Opening and Total and Maximum Shipments of Iron-Ore by the Various Iron-Ore Ranges in the Lake Superior Region to the close of 1886.

Range.	Commenced shipping ore.	Total shipments to close of 1886.	Maximum annual shipment to close of 1886.
		Long tons.	Long tons.
Marquette,	1856	23,346,819	1,829,394
Menominee,	1877	6,196,687	1,136,018
Gogebic,	1884	877,069	756,237
Vermilion,	1884	592,004	307,948
Miscellaneous,		48,432	
Total for Lake Superior Region,		31,061,011	3,565,144*

At the date of the Duluth meeting the actively producing iron-ore territory of the Mesabi range, which last year contributed over three million tons of iron-ore, was an unbroken wilderness—as much so as the areas now recognized as the Gogebic and Vermilion ranges were, at the time of the Institute's first visit. The first shipment from the Mesabi range was made in 1892, and it has become a most important factor in the supply of ore from this region. For comparison with

* The maximum annual shipment for the whole region is, of course, not the sum of the maxima of the several ranges.

the foregoing statements the total shipments and maximum annual output of the various ranges to the close of 1896 are given below.

Year of Opening, Total Shipments to close of 1896, *and Maximum Annual Output of the Lake Superior Iron-ore Ranges.*

Range.	Commenced shipping ore.	Total shipments to close of 1896.	Maximum annual output to close of 1896.
		Long tons.	Long tons.
Marquette,	1856	46,538,187	2,863,848 (1890)
Menominee,	1877	22,994,428	2,402,195 (1892)
Gogebic,	1884	20,788,787	3,058,176 (1892)
Vermilion,	1884	9,220,235	1,226,220 (1892)
Mesabi,	1892	8,074,583*	3,082,973 (1896)
Total,		107,616,220	

Attention is called to the fact that in the seven years between the first and second visits of the Institute, the total shipments of Lake Superior ore were greater than the aggregate for the twenty-five preceding years, and that in another interval of ten years the Institute finds the total output for the decade 1887 to 1897 is more than double the quantity won in all previous exploitations.

The Mesabi range has developed more rapidly, has had a greater annual output than any of the other ranges, and apparently takes the lead in the amount of ore-reserves. Conservative estimates made of the explored and developed properties indicate that there is practically "in sight" in the Mesabi range double the amount of iron-ore that has been shipped from the entire Lake Superior region during the past forty years of active development.

The present status of the various ranges, as indicated by their output in 1896, is as follows:

Range.	Long tons.
Mesabi,	3,082,973
Marquette,	2,418,846
Gogebic,	2,100,398
Menominee,	1,763,235
Vermilion,	1,200,907

We are accustomed to make comparisons by decades to obtain yearly averages, but equally fair results should follow

* 5 years' shipments.

comparisons based upon the intervals between the three visits of the Institute, if the fact is kept in mind that the Menominee district had been shipping ore but three years at the time of the first visit, that the Gogebic and Vermilion districts had been active but three years at the time of the second visit, and that the Mesabi range is now working on its sixth year.

Average Annual Shipments of Ranges.

Range.	Total. From close of 1879 to close of 1886. Long tons.	Total. From close of 1886 to close of 1896. Long tons.	Average. 1880-1886, 7 years. Long tons.	Average. 1887-1896, 10 years. Long tons.
Marquette, .	10,711,349	23,191,366	1,530,193	2,319,137
Menominee, .	5,821,452	16,797,741	831,636	1,679,774
Gogebic, . .	877,069*	19,911,718	292,356†	1,991,172
Vermilion, .	592,004*	8,628,231	197,335†	862,823
Mesabi, . .	none.	8,074,583‡	.....	1,614,917§
Total and average for Lake Superior region, .	18,001,874	76,603,639	2,571,697	7,660,364

From these statements it appears that the older ranges have not been crowded out of business by the later additions, but rather that the increased demand has been met by the newer ranges. The competition which followed the opening of new mines has demanded that economies be practiced by all, and much of the ore which has been supplied from the younger ranges could have been furnished by mines opened before these districts came into notice. The figures, however, demonstrate that each range is evidently making the most of its natural advantages, and endeavoring to discount its drawbacks, whether these are in the character or quality of ore, the distance of rail- and water-transportation, the method of mining, or the charges for interest on capital invested or for royalties.

Although some of the mines which have been important producers show signs of depletion, in a number of cases new finds have increased the apparent reserves; we may, therefore, conclude that each of the five ranges now embraced in the Lake Superior region will continue to supply ore, even if as great surprises await the future explorer as have startled the metallurgical world. The commendable efforts of the mining

* Total for 3 years. † Average for 3 years.
‡ Total for 5 years. § Average for 5 years.

engineer, the mechanical engineer, the civil engineer and the marine engineer have made the present remarkable development of the iron-ore mines of the Lake Superior district possible; and while the contest for local supremacy has been severe, its results have affected mines in other sections of the country even more seriously than on the various ranges of this region.

Economical mining and handling, and low charges for transportation, have extended the market of the Lake ores close to the shaft-houses of many mines producing ore in small amount or of relatively inferior grades; and, as a consequence, the proportions which the ores mined in the Lake Superior region have borne to the total production of the United States has increased until it may be said that, in round numbers, 5 tons out of every 8 obtained from domestic mines come from the Lake Superior region.

Many of the mines in the Eastern and Central States which have succumbed to competition have resources of ore which may again be called for, if methods of extraction and beneficiation can be so cheapened as to permit of the mineral finding a profitable market. There are also known deposits of iron-ore of such quality and extent as to offer strong encouragement for the future, when the demands of the middle and western portions of the country make their exploitation possible.

Moreover, there are liberal supplies in the Southern States which have supported, and which will continue to support, an important iron-industry. We may, therefore, feel assured that, in the light of present metallurgical knowledge, the United States is well supplied with iron-ores.

Within the last decade the proportion of pig-iron classed as Bessemer has been increased from 40 to 60 per cent. of the total production, which (after allowing for the average comparative leanness of non-Bessemer ores utilized) would indicate that from 30 to 50 per cent. of the weight of domestic ores used has been of Bessemer grade. Although the limitations between Bessemer and non-Bessemer ores have been more closely defined in late years, the growth of the open-hearth steel manufacture has done much to relieve anxiety as to the future supply of ore for the steel trade. But if such were not the case, the abundance of ores with high percentages of iron, but with phosphorus above the present Bessemer limit, which

can be cheaply mined, would encourage experiments which will lead to their utilization. Much as iron and steel metallurgy has advanced, there are many improvements yet to be made, and with an army of technical and practical minds studying possible improvements, one hesitates to make prophecies.

The abundance, in the Lake Superior region, of ore of excellent character, but beyond the phosphorus-limit for Bessemer purposes, offers the expectation that pig-iron for other than Bessemer use will continue to be produced at low cost with the aid of cheap mining and transportation.

The writer lately had opportunity to examine the labor accounts for three consecutive months connected with the mining of coal, coking of the same, mining of ore, transporting raw materials to furnace, the production of pig-iron and the delivery of the same on cars. The total number of days' labor at mines, ovens, furnace, and a proper proportion for railroad transportation, divided by the total tons of pig-iron produced, gave a quotient less than 4. That is, pig-iron is now being produced in the United States for less than four days' labor per ton of metal, from the time the native materials are attacked underground to the delivery of the pig-metal on cars for shipment. Such is the competition which the miners of ores must be prepared to meet. The books of another blast-furnace plant showed that 3½ tons of pig-iron were produced for each day's labor connected with the furnace. In each of the above instances every employee outside of the office-force was included in the total; and while each record may be considered as exceptional at the present time, they set a standard which all engaged in the production of pig-iron must expect to meet. Those who supply ore, as well as those who smelt it, must be able to reduce their labor-costs to a minimum.

The development and present status of the most important contributor to our domestic iron-ore supply having been indicated, attention may be called to the advances made elsewhere to meet the inroads of Lake Superior ores into what were assumed to be local markets. The older iron-producing districts of the United States, in contending for a share of business, have cheapened the exploitation, and in some instances improved the quality of local ores; so that it is not to be expected that the proportion which the Lake Superior ores bear

to the total annual output will be greatly augmented in the immediate future.

In the eight years for which statistics of domestic iron-ore production have been collected under the authority of the general government, the output ranged from eleven and a half to sixteen million tons, the average for the eight years approximating fifteen million tons. The average value per ton of ore at the mines, however, has shown a marked decline. This value, in the years named, was as follows:

In 1889 the average value per long ton of ore was	$2.30
" 1892 " " " " "	2.04
" 1893 " " " " "	1.66
" 1894 " " " " "	1.14
" 1895 " " " " "	1.14
" 1896 " " " " "	1.42

The " ore-pool " maintained prices for much of 1896; hence the apparent advance in that year.

Taking as the normal demand for domestic ores the average annual production of the iron-ore mines of the United States in the past eight years, viz., 14,670,000 long tons, and making allowance for the prevailing industrial depression, and for the growth attributable to increase of population, and also for the general demand for ores carrying high percentages of iron, it is doubtful if the interval between January 1, 1897, and the end of the century will require for actual use more than 53,000,000 long tons of iron-ore, or that any new discoveries will be required to supply this amount conveniently, unless some unusual advance takes place.

The iron-ore supply is not confined to domestic mines. The United States has drawn upon Spain, Italy, Cuba, Algeria, Sweden, Great Britain, Greece, Canada and British Columbia, and in view of the importations which have been or may be made, a general statement of the ore-supplies of other countries may be appended.

The importation of iron-ores in late years has not been excessive, having ranged from 1,246,830 long tons in 1890 to 167,307 tons in 1894, the total for 1896 being 682,806 long tons; but an increase in the quantities brought into this country may be expected if the conditions favoring such traffic are brought about by advancing prices here, without similar improvement elsewhere.

The nations which are important producers of iron, or which have known deposits of iron-ore of magnitude, may be scheduled as follows:

Great Britain, in earlier times, depended almost entirely upon native iron-ores; but the introduction of the Bessemer process, requiring ores of special grade, has necessitated large importations of foreign ores, reaching nearly four and a half million tons in 1890, and in late years averaging about four million tons. The present domestic production is about eleven and two-thirds million tons.

Germany produces the greater proportion of the iron-ore smelted in the blast-furnaces of that country, and also exports a large amount, about two and a quarter million tons, principally to France and Belgium. It also imports about one and a half million tons of rich foreign iron-ore, chiefly from Spain, Sweden, etc., for use in steel manufacture. The total home production is twelve and a third million tons.

France formerly mined all of the iron-ore which was used in her blast-furnaces; but after the loss of the provinces of Alsace and Lorraine, which contain large deposits of oolitic iron-ore, the importation of ores, principally from Germany, Spain and Algeria, increased, until nearly one-third of the supply comes from foreign sources. The domestic production is about three and a half million tons, and the importation is one and two-thirds million tons per annum.

The Austrian empire mines somewhat over two million metric tons of iron-ore per annum, supplying practically all the ore needed for its blast-furnaces, so that the exports and imports nearly balance each other.

Russia has large deposits of iron-ore and easily supplies the iron-ore necessary for her increasing pig-iron output, the amount now mined being about two million tons per annum.

Spain possesses fine iron-ore deposits, some of the product of which is smelted in Spanish blast-furnaces, but the bulk of the five and a half million tons produced is exported to foreign countries, principally Great Britain, Germany, France and the United States.

Sweden mines all the iron-ore smelted in that country, and since the exploitation of the rich deposits in the northern section of the country, has become a large exporter of iron-ore,

chiefly to England, Germany and the United States. The output in 1895 was one and nine-tenths million tons, of which probably slightly over one-third was exported.

The Chinese empire uses the iron-ore which it produces, none being exported; and owing to the crude methods employed, it is estimated that one and a half million tons of iron-ore are annually required to meet the demands of a limited iron-manufacture.

Belgium, which at one time produced practically all the iron-ore required by her blast-furnaces, has been compelled by the depletion of her deposits to import large quantities of ore (about one and a half million tons annually); the domestic output being usually about two hundred thousand tons.

Cuba has good iron-ore resources, but all the iron-ore mined is exported, the amount annually shipped to the United States (which consumes all the ore produced) being about one-third of a million tons, under ordinary conditions. The mines on this island are practically owned and worked by United States capital.

Algeria has also iron-ore reserves, and, like Cuba, ships all the iron-ore mined, which amounts to about 400,000 tons annually, the principal markets being France, Great Britain and the United States.

Italy ships much of the iron-ore mined, the island of Elba producing the bulk of the ore. The exports have decreased in late years, and at present less than 200,000 tons are shipped.

Greece exports all the iron-ore produced, the amount so shipped being slightly less than 150,000 tons in 1895.

Canada has good deposits of iron-ore, and in addition to supplying its own wants, occasionally sends small amounts to the United States. The annual production is about 125,000 tons.

India and Australia also possess large deposits of iron-ore; but these have not been as yet fully exploited. The former country, however, supports a limited iron-industry, which is estimated to require about 50,000 long tons of iron-ore annually.

Mining Methods on the Mesabi Range.

BY C. E. BAILEY, VIRGINIA, MINN.

(Lake Superior Meeting, July, 1897.)

IN this our fifth year of actual mining, a *résumé* of our methods of mining may prove of interest.

Our conditions are radically different from those of other districts; and we are just commencing to be able to draw conclusions regarding the systems in use.

The ore-bodies of the range lie in a blanket-formation varying in width from 200 or 300 feet to over half a mile, and in length from 500 feet to more than 2 miles. It is practically horizontal in bedding, but in places has a dip of as much as 7 feet in 100 for a quarter of a mile. The top of the ore is in most cases nearly level, although local rolls make the back (or top of the ore) vary to the extent of 10, or in rare cases 20 feet, and very frequently there is a general slope of the top of the ore, as well as the bottom, which makes it impossible to mine it all from one level. Taking the average of long stretches, the side-walls may be described as quite regular; but local deflections make it necessary to prospect for the wall-rock, and in drifting underground it is common to drive along the side-walls, uncovering them almost continuously, and thus making sure that no large pockets of ore are missed. The dip of the side-walls varies greatly, but an average is probably between 45° and 60°, sloping inward at the bottom, making the general shape of the ore-body not exactly a basin, but more nearly a trough, usually with a longitudinal trend and sloping sides, and ore much the deepest at the center. As to rock-occurrences inside the ore-body, the properties most largely worked may be said to be particularly free from rock, although much trouble is occasioned by bunches or small masses. In most cases these are not properly described as "horses" of rock, since they have neither continuity nor regularity of trend, but are offshoots connected with the side-walls or bottom.

The depth or thickness of the ore varies from a few feet to over 300 feet, but in rare instances exceeds over any large area 150 feet of continuous high-grade ore.

A depth of 60 feet of uniform good ore makes the mining problem fairly simple; and in some instances a depth of 35 feet is mined cheaply.

As a general rule, the top-ore is poorer in quality, being especially high in phosphorus; and the same may be said of that near the side-walls, bottom and ends of the deposit, except that in these cases the percentage of iron is usually low as well. The best ore is in the heart of the deposit, and this mode of occurrence has led to a comparison with a watermelon.

The ore is a soft hematite, but varies greatly in hardness in the different districts. The Hibbing district (especially at the Mahoning end) has a remarkably soft ore, capable of being drilled with an auger or loaded with the steam-shovel without blasting. The Mountain iron-ore is but slightly harder; while the Virginia deposit has ribs of very hard ore, and the Eveleth formation is by far the hardest on the range.

The surface consists of glacial drift from 10 to 150 feet in thickness. It is clay, well filled with granite boulders, except the last few feet above the ore, which is sand and gravel with some small boulders. It is an exceedingly hard material to handle. Most of it would be classified as hard-pan.

The side-walls and underlying rock are taconite, and the change from clean ore to rock usually occupies but a few feet, although cases are not infrequent in which 10 to 30 feet of low-grade ore, too poor to be merchantable, intervenes.

The two systems of operating are the stripping or open-pit and the underground system.

The stripping-mines are worked either by steam-shovel or the milling-process. Much that is to be said applies equally to both.

In determining whether a property should be stripped or worked underground, there is still a decided difference of opinion regarding the depth of surface, either absolute or relative, which constitutes the dividing line.

Originally the proposition was that a foot of surface could be stripped for every foot in depth of the ore; but, while the figures indicate that this is theoretically safe, I do not think any

manager would to-day put it in practice. Whatever change has taken place is in the direction of less stripping and more underground work.

There are instances of stripping 55 feet thick in individual places, and of a general average over the whole ore-body of more than 35 feet; and nowhere is the average over any large area as low as 20 feet. Most estimates on the amount of stripping are grossly inaccurate because of the large allowance necessary to cover berm, slopes and approaches. Just how much to allow for approaches, in order to steam-shovel the deeper ore, is not yet fully apparent. As to relative depth of ore and surface, it is probable that 1 foot of surface to 2 feet of ore would nearly constitute the dividing line in actual practice to-day.

If the average thickness of ore were less than 70 feet and the average surface-layer exceeded 35 feet, it is likely that most managers would select the underground system. The fact that market-conditions have made the probable output of a property a matter of the vaguest guesswork has operated to make managers very careful about heavy preliminary outlays.

The benefits of stripping are many and great, and the saving in mining cost is large. The absence of timbering; the use of black powder instead of dynamite, and the heavier charges of powder; the possession of daylight and pure air; the greater safety; the possibility of immense outputs just when desired; and the almost entire absence of expense outside of the shipping-season, make this plan of work the ideal one for the operator, but the most undesirable for the laborer and for the town near which the mine is situated. All the benefits mentioned accrue equally to the milling and the steam-shovel systems. The average cost of stripping prior to 1897 has not been much less than 50 cents per yard, while the present cost is between 30 and 35 cents per yard. There is to-day a marked improvement in handling the earthwork, not in general methods but in the use of powder and a better acquaintance with the behavior of the material. In one instance, after 3 years' experience, an operator moved as many cubic yards in a month with one shovel as had been done at first with three.

In stripping, a simple inspection will usually determine what ore is up to grade; but in many instances analyses and careful

calculation are necessary to determine whether the loss would be greater to ship as ore or cart off to the waste-dump as stripping. As to cost of stripping per ton of ore, it is probably safe to name 10 cents per ton of ore as a minimum and 20 cents as a maximum. In this estimate the original cost has been augmented by a reasonable allowance for interest on investment, as it would naturally be some years after ore was stripped before it was all mined.

A description of the steam-shovel work will not be given here, as it involves only problems of earthwork handling, and is only indirectly connected with mining proper, as fixing costs with which the latter must compete.

The milling-system consists of stripping the ore and hoisting through a shaft the ore which is worked in an open pit and "milled" into underground cars. The location of the shaft, the depth at which the tramming-level is started, and the plan and size of drifts are the same as in underground mining, and will be described in that connection.

After the shaft is sunk and the drifts are completed, raises are put up to the top of the ore about every 50 feet, and chutes are put in at the level. Mining then begins at the top, by drilling beside the chute 8 to 15 feet deep, and, after squibbing with dynamite, loading the hole with black powder and firing. In most of the Mesabi mines the ore breaks up sufficiently fine in falling to "mill" through the chutes; but in the Adams mine it is so hard that great difficulty is experienced from the tearing out of the chutes by the falling masses.

This system has all the advantages of the steam-shovel method (unless it be in the matter of pumping) and the additional advantage that no change in system is required as greater depth is reached, and also that no long approaches are necessary. The cost probably exceeds that of steam-shovel work by about 10 cents per ton of ore. In open-cut mining it is possible to take out all the ore, while no plan of underground work can avoid losing a considerable percentage. In mining soft ore with a sand capping, the taking care of the hanging-wall is the expensive part; and if this is removed entirely, the benefit accrues to all the ore uncovered, irrespective of method of operating or the depth at which mining is done. Thus if, with greater depth, loading with steam-shovel into railroad-cars

should not prove feasible, the benefits of having uncovered the ore do not on that account cease.

In underground work on the Mesabi range, the planning is greatly simplified by the possibility of determining in advance the limits of the ore. This is accomplished by test-pits sunk until stopped by water, and the remainder tested by drilling. The cost is from $1.25 to $3 per foot until solid rock is reached; and by sinking a pit every 250 feet each way, the top-, bottom- and side-walls are determined as well as the quality. If there has been any change in the practice of operators, it is in the direction of more thoroughly prospecting before commencing to equip for active mining. In selecting the location of a shaft, it is desirable, if possible, to place it in or near the deepest ore; to plan that the tracks shall pass over as little ore as possible; and to have the possibility of a 1 per cent. grade down to the shaft as well as away from it.

Where it can be accomplished without too great inconvenience, the shaft is placed near the rock-wall, and as central as possible, to avoid long trams. It is customary to avoid sinking through rock, because of the expense; but this is of less importance than the preceding conditions, since it would add less than $1500 of total expense; and this, with interest, would be less than 0.5 cent per ton on 500,000 tons. When a new shaft is to be sunk, it is the preference of the writer to sink a test-pit through the surface near the proposed location of the shaft and drift on the top of the ore or rock until the junction is reached, and then sink the shaft on a dip of 70° with foot-wall 3 or 4 feet from rock. The choice between vertical and incline is largely a matter of taste; but 70°-inclines have seemed to have the preference lately. Our shafts all have three compartments, of which two are for hoisting and the third for pipes and ladder-way; but there is a tendency developing to avoid fire-risk by placing the ladder-way away from the hoisting-shaft. In most cases, the bottom of the shaft is provided with a pocket sufficient to hold a few skip-loads, to avoid delay in case of a temporary stoppage of the skip.

In selecting the depth at which to start the main tramming-level (the ore having been previously well tested by pits), it is usual to place it as near the bottom of the ore as possible, unless two 50-foot stopes can be planned for; in which case the

lower one is placed at the bottom of the deposit, care being taken not to go so low that rock-drifting would be necessary. Where possible, a height of about 50 to 60 feet above level is preferred. Near the shaft, the main gangway is usually 14 feet wide within timbers; but away from the shaft 12-foot caps, giving 10 feet in the clear, are common.

There are two distinct systems of drifting, in one of which 12-foot drifts with right-angled turns are used, while in the other all turns are of 45°, and, with the exception of the main gangway, the drifts are 5 or 6 feet wide and untimbered.

The two general methods of stoping in use on the Mesabi range are the subbing-and-slicing system and square-set work. A third system (the use of saddle-back stulls) which, in the opinion of the writer, promises a reduction in cost over either of the others, is still in the trial-stage.

In the first, sub-levels are planned every 12½ feet above the main level until the top of ore is reached, and the top sub is worked out, planked and caved, and each below is taken in turn. By this system all ore is drifted out and shoveled and dumped into chutes leading to the main level, where it is trammed to the shaft. The method is based on the principle that drifting gives the cheapest ore.

By the square-set method, 40 per cent. or more can be "milled" into cars, and, by overhand stoping, the product per man in the room is made large; but in filling the room and taking out the intervening pillars the expense is greatly enhanced. In a year from now more definite information can be had regarding the removal of pillars between square-set rooms.

In the third system, a 16- or 20-foot drift is run next the sand or top of the ore, and another directly under it on the main level. The top one is timbered by cutting hitches in the side-walls and setting in stulls, which meet at the center and brace against each other like the rafters of a house; and over these stulls is placed lagging, up to the back. A raise is then put up, and underhand stoping begins from the top. When the room is ready for stoping, the product per man is surprisingly large. The plan has but lately been put on trial, and several problems remain unsolved. Whether it is best to drive the top drift first; whether to put up the raise at the inner or outer

end or the middle; whether any or nearly all the ore from the room can be "milled;" whether the room will be harder to fill than a square-set room; whether the pillar can be removed as readily as in square-set work; and whether the bottom drift ought to be of the full width of the room or just wide enough for one car—all these problems will be studied and thought out, and in another year more definite information ought to be available. For the introduction and development of this system great credit is due to Mr. Geo. H. Wallace, superintendent of the Fayal mine. This is not the first time Mr. Wallace has shown himself one of our most progressive mining men in planning new methods.

In economy of ore the first method seems to the writer to have the advantage, while in cost per ton the third and second seem to have the preference. The best examples of the first method can be seen at the Genoa and Fayal mines; of the second, at the Lake Superior group, and of the last at Fayal.

Our tramming is all done by hand, except at Fayal and Norman, where mules are used. It is not likely that electric or rope-haulage will be considered for our range, since long-distance tramming is now being done at rates that compare favorably with these systems, and long trams are not a necessity with us because, with our shallow depth, a new shaft can be sunk and equipped for $6000, which, on 600,000 tons, would be only 1 cent per ton. With market conditions so unstable and the probable output so uncertain, as they have been ever since our range started shipping, I think most of our managers would hesitate to install a costly plant unless the benefits exceeded what now seems apparent. I do not mean to imply that our mines are worked with insufficient capital, or that our managers lack breadth of view or foresight. In both respects the exact opposite is true. We have no apologies to offer for our mining; and if, after inspecting our mines, the members of the Institute have anything but words of praise for the systems introduced by Mr. Bacon and Mr. Olcott, and the business-like way in which they are carried out to the smallest detail, the writer will be disappointed. It is only intended to suggest the reasons why heavy preliminary outlay is avoided wherever possible.

A description of methods of sampling will not be given, be-

cause there is no uniformity of practice. The exigencies of each mine determine the underground or pit-sampling, while shipments are sampled in 20 car-lots, except at Mahoning, where every 5 cars yields a sample.

Our difficulties, records, costs and prospects will receive but short mention.

The manager may have several drifts in rock and still greet his friends pleasantly; he may have heavy fixed pumping-charges, or a temporary deluge, and learn to face the situation with equanimity; he may lose a shaft or building by fire and recover from the loss; but what keeps him awake nights is trying to plan ahead without the least possible clue regarding what tonnage may be called for.

In underground work, the Fayal record of 57,000 tons in a month surpasses any single shaft-record I have happened to hear of. This was hoisted with skips holding a little over 3 tons. A milling-mine, the Auburn, with a 6-ton skip, hoisted 67,000 tons in a month.

In steam-shovel work, the Mountain Iron records of 5800 tons in 10 hours and 164,000 tons in a month, are unsurpassed, but will probably be exceeded very soon.

Regarding costs, it is unwise to say more than that the milling-system probably puts ore on cars for about 55 per cent. of the underground cost, if stripping is included, while the total steam-shovel cost is probably nearer 33 per cent. of the underground.

Regarding the future, it seems unsafe to predict much cheapening in mining cost, as economy is already practiced on every hand, the mines are equipped in a thoroughly business-like way, and the wage-scale ought to rise rather than fall. A few improvements can be made; but costs are about as low as ought to be expected, and managers may take some pride in having reduced them to the point they have reached.

Explorations on the Mesabi Range.

BY E. J. LONGYEAR, HIBBING, MINN.

(Lake Superior Meeting, July, 1897.)

THE rapid development of the Mesabi range has been a matter of much wonder to those familiar with ore-deposits elsewhere, and much skepticism was shown when "millions of tons of ore on the Mesabi" was first reported. Now, however, with over 8,000,000 tons to its credit at the close of 1896, a prospect of 3,000,000 more this year, and more ore "in sight" than ever before, it is evident that the first claims for the range were not so greatly exaggerated as perhaps even those who made them thought they were at that time. An explanation of the ease with which these deposits have been explored is found in their mode of occurrence. Lying in flat beds and near the surface, and only exceptionally occurring under a taconite "capping," the prospector had only to sink a few vertical test-pits on a forty-acre tract down to the rock to be reasonably sure whether his property contained ore or not. If he found ore in one or more of these preliminary pits, he would then fill in with more pits until the ore-area was entirely outlined by test-pits from 200 to 400 feet apart. In the early days of the range the testing was almost entirely done by digging pits with pick and shovel. When water was encountered, work had to be stopped, unless a steam pumping-plant was put in, resulting in greatly-increased expenses. Latterly, drills have been used with great success in continuing the exploration after water has been struck in the pit, rendering further progress with pick and shovel impracticable.

There are several kinds of drills in use. The diamond-drill is largely used, and possesses the advantage of being in readiness to drill rock if thought advisable, the taconite being very hard and almost impossible to drill except by the use of diamonds. Some form of churn-drill has been much used. This may be either a steam- or a horse-power-drill. The principle is the same in each, the cutting being done by percussion instead of by rotation, as in the case of the diamond-drill. A churning motion is given by passing a rope from the top of the

drill-rods up through a sheave-wheel and down again, running two or three times around the drum of the engine, leaving the end loose. The drum continually rotates, and a man at the loose end alternately tightens and slackens the rope, thus raising and dropping the drill-bit at will. The bit itself usually has a single cutting-edge, rather thin toward the lower end, and a thread cut on the upper end, in order to attach it to the drill-rods, which may be the flush-joint rods used with the diamond-drill or extra-strong gas-pipe connected by extra-long couplings, to protect the threads from the strains occasioned by the pounding of the bit. In surface-drilling a 3-inch casing of common gas-pipe and a 1¼-inch drill-rod are used. In ore, a 2-inch casing is put in after the 3-inch has been stopped, and a 1-inch drill-rod is used. A 4½-inch casing-pipe has been tried several times for deep surface-work, and after it could be driven no further, a 3-inch pipe was inserted; but it was found that the sand is quite liable to rise between the two pipes and hold them together, preventing the smaller pipe from being driven. As a rule, it is safer to start from the top with a 3-inch pipe, and make every effort to carry it through the surface.

Surface-drilling requires more experience on the part of the drill man than ore-drilling, which is simpler in operation, although requiring great care in order to secure reliable results. In surface-drilling many difficulties arise calling for much ingenuity. A bit may be lost off and be wedged across the hole in such a position that it cannot be recovered, when perhaps a stick of dynamite will force it to one side, out of the way. Sand may rise between the drill-rods and casing-pipe at a time when the pump refuses to work, holding the two together so tightly that jack-screws will not loosen them. A number of methods will probably have to be tried before the rods can be recovered. A boulder of any size is best overcome by shattering it with 60-per-cent. dynamite, care being taken to raise the casing-pipe at least 5 feet after lowering the dynamite to the top of the boulder. An electric battery is used to do the blasting. Two or three blasts will allow the casing to pass through almost any boulder unless it is an unusually large one. Careful blasting and attention to keeping the pipe loose by turning it while driving are important points to be observed.

The cuttings are washed out by means of water forced down by a pump through the hollow rods and up again between the rods and casing. The operations of drilling, washing-out, and driving the casing are carried on sometimes singly, sometimes two at once, and sometimes all three together, according to the nature of the ground. A No. 4 pump is a good size to wash with. It can be easily lowered into a small test-pit, and it will handle enough water in most cases. A cast-iron cylinder 10 inches in diameter and 12 inches long, with a 2-inch hole through the middle, weighing 240 pounds, makes a good drive-block. The hole is for the rods to pass through and act as a guide for the block, which is raised by attaching the drill-rope to it and operating it the same as in raising and dropping the drill-rods.

After striking ore, if the material just above it is sandy, it is better to drive the 3-inch pipe 5 feet into the ore and stop, continuing in ore with a 2-inch pipe, as fine sand is apt to run down behind the casing and injure the ore-sample if the same pipe is used in sand and ore. The Mesabi ores are too soft to make a core with a diamond-drill, and are generally sampled by the churn-drill, the cuttings made by the bit being washed out through a tee in the top of the casing. Even when a diamond-drill is used it is better to run it as a churn-drill, although the more rapid progress made with a diamond-bit offers a temptation to use it. But the revolving rods, grinding on the casing-pipe, wear off a large amount of metallic iron, which mixes with the ore-sample, producing, at times, surprising results in the assays. The cuttings made by the chopping-bit are often washed out at the same time that the drill is working; but more accurate results can be obtained by drilling "dry," that is, by chopping without pumping any water through the rods, the cuttings mixing with the water that stands in the hole. When the depth desired to be sampled has been drilled, the casing-pipe driven down, and the material that has been packed by driving loosened by re-drilling, the chopping is stopped and water is turned on, flushing the sample to the top into barrels. The pump should be worked to its fullest capacity in order to bring up the complete sample with a minimum amount of water. One barrel of water will usually be sufficient to wash out a 5-foot sample unless the hole is more

than 200 feet deep. After settling until the water is clear, it is drained off, and the sample is taken out, dried, quartered down if desired, sacked and properly marked. When the ore is drilled "wet," the water carrying the sample is run through a series of three or four settling-barrels, the overflow from one passing into another. When a sample has been drilled, the clear water is drained off, and the sediment in the different barrels is collected together and dried.

The dry method of drilling is preferable to the wet for two reasons. First, it drills a hole of uniform size, a very desirable feature in accurate sampling, whereas in the "wet" method a stream of water is continually playing around the bit and will wash out more of the soft layers than the hard. The Mesabi ores usually occur in alternate layers of hard and soft ore. Secondly, the water overflowing the barrels is liable to carry away more or less of the lighter parts of the sample, making it too rich in iron.

The writer made some tests to determine the effect of incomplete settling on the sample, and also the effect of heat in drying. A sample was made up by mixing a lot of blue ore, brown ore, yellow ore and paint. After crushing to about the fineness of a drill-sample, it was carefully mixed and divided into five portions. These were treated as follows:

Portion A was analyzed for iron and phosphorus; B was mixed in a pail of water, allowed to settle sixteen hours, water poured off and sediment dried at less than 212° F.; C was mixed in a pail of water, allowed to settle sixteen hours, water poured off and sediment dried at high temperature; D was mixed in a pail of water, settled sixteen hours, sediment dried with considerable heat until nearly dry, and then finished slowly; E was mixed in a pail of water, settled twenty minutes, water drained off and sediment dried at less than 212° F.

The analyses of these dried ores resulted as follows:

	A.	B.	C.	D.	E.
Iron, per cent., . .	59.23	59.69	59.95	59.67	60.80
Phosphorus, per cent., .	0.087	0.086	0.087	0.088	0.094

The conclusions to be drawn from this test are that too much pains cannot be taken in allowing a thorough settling of the ore-sample before removing from the barrels, and that a rapid

drying will not injure the sample materially. I have, however, made a practice of using considerable care in drying.

The taconite underneath the ore is hard to drill, but with the aid of dynamite enough can be secured to determine its character.

The drilling on the range is mostly done by contract; $2 to $2.50 per foot being paid for ore, $2.50 to $3 per foot for surface, and $4 to $5 per foot for taconite. Three men to each crew work to the best advantage. Wages are from $1.35 to $2.25 per day. A good outfit costs about $800.

An accurate map of the work should be made, drawn to a scale of 200 feet to the inch, and the ore-area marked on it by a line inclosing all the ore-pits. It is the practice to allow 100 feet outside the ore-pits in locating this boundary line. The ore-area being measured on the map by means of a planimeter is reduced to square feet and multiplied by the average thickness of ore as determined by the different pits, giving the contents in cubic feet probably as nearly accurate as the prismoidal formula would give it. To reduce this to tons, 13 cubic feet is usually taken as equal to one ton.

The Marquette Range.—Its Discovery, Development and Resources.

BY JAMES E. JOPLING, MARQUETTE, MICH.

(Lake Superior Meeting, July, 1897.)

THE county of Marquette, Michigan, includes nearly all the iron-mines that have been worked on the Marquette range, which stretches in a generally western direction from the mines at Negaunee, 10 miles from Lake Superior, to those beyond Michigamme, some thirty miles further. The geographical position of this range, nearer to eastern ports than the other iron-ranges, and its ready access from the lake, gave it for a long time a controlling advantage. The country in which the mines are situated is from 800 to 1000 feet above the level of the lake, and is for the most part broken by steep hills, which rise from 100 to 200 feet; between these the land is rolling or

swampy. The eastern part is on the water-shed between Lakes Superior and Michigan, affording good drainage.

Geologically, the range is in the Algonkian area, which is here generally less than 10 miles wide, and is bounded north and south by the Archæan, consisting of syenite, gneiss, granite, etc. The Lower and Upper Marquette formations, into which the Algonkian has been divided by Dr. Van Hise, of the U. S. Geological Survey, correspond to the Huronian formation of Canada. It is in a sub-division, termed by the same geologist the Negaunee formation, that most of the ore-bodies occur, generally associated with the greenstones, which in many places underlie them, and in others cut across the formation in the form of dikes. Generally speaking, the soft hematite ores occupy a lower geological horizon than the hard ores, which, in some places, are found to rest unconformably upon them.

The ore-bodies are extremely irregular in shape and size, and appear to be segregations in the mass of jasper, with which they are always associated. As a general rule, the soft ores occur beneath the jasper, and either in complete or partial basin-shaped deposits, in synclinal folds of the greenstone or schist, with a pitch to the west. The hard ores, on the other hand, usually occur above the jasper and next to the overlying quartzite, etc., of the Upper Marquette formation. The depth to which these formations extend is yet unknown, although in the oldest mines which contain the hard ores from 1200 to 1300 feet has been reached. The ore at greater depth is not as extensive as near the surface. Soft ore has not been followed to such a depth on this range; the deepest mine of this description reached nearly 1000 feet from the surface before the various shoots or chimneys from the main mass pinched out. The Figs. 1 to 11, inclusive, were prepared by the U. S. Geological Survey, and show the characteristics of the ore-deposits in this district. I am indebted for them to Mr. George A. Newett, who is publishing them in his *Annual Report*, as Commissioner of Mineral Statistics of Michigan.

Fig. 1 is a generalized section, showing the relations of all classes of ore-deposits to associated formations. On the right is soft ore, resting in a V-shaped trough between the Siamo slate and an eruptive dike of soapstone. In the lower central part of the figure the more common relations of soft ore to

vertical and inclined dikes cutting the jasper are shown. The ore may rest upon an inclined dike, between two inclined dikes, and also upon the upper of the two, or it may be on both sides of a nearly vertical dike. In the upper central part of the

FIG. 1.

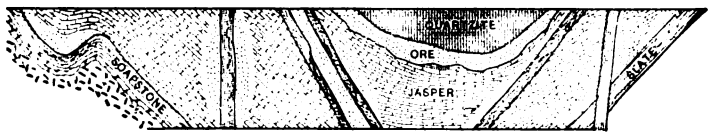
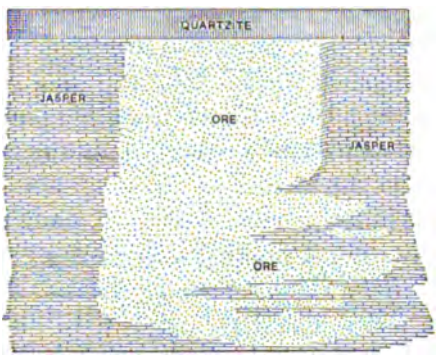


FIG. 2.

FIG. 3.



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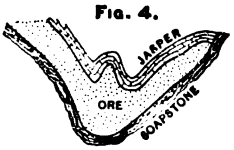

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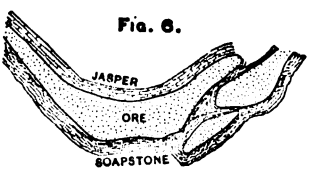

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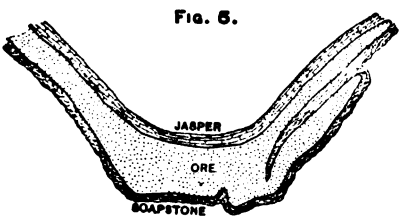


FIG. 7.

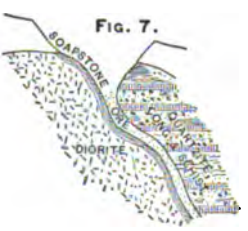


figure are seen the relations of the hard ore to the Negaunee formation and the Ishpeming quartzite. At the left is soft ore resting in a trough of soapstone which grades downward into diorite.

Fig. 2 exhibits sharply-plicated jasper (black belts) and ore (white areas), showing the shattering of the jasper and concentration of the ore. The ore is proportionately greater where the folding has been sharpest. This figure was drawn from a photograph from the southeast corner of the Republic horseshoe.

Fig. 3 is a horizontal section of a chimney of ore on the east side of the Republic horseshoe. The left side of the ore is bounded by cross-joints. The right side is bounded in part by a sharp flexure, passing into a joint, and in part grades into the lean banded jasper and ore. The scale is 20 feet to the inch.

Figs. 4, 5 and 6 are three cross-sections of ore in a trough of soapstone grading downward into diorite. In Fig. 4 the ore-deposit is solid. In Fig. 5 a dike shoots off, and nearly separates this ore-body into two parts. In Fig. 6 the two dikes divide the same ore-body into three parts. This scale is 200 feet to the inch.

Fig. 7 is a cross-section of the National mine. On the left is soapstone, grading into diorite. Above this is hard ore; and overlying the hard ore are interstratified conglomerate, quartzite, and schist. The ore is here plainly due to a replacement of the silica of the different sedimentary bands by ore, although the original conglomerate was heavily ferruginous. The scale is 200 feet to the inch.

Fig. 8 is a cross-section of Section 16 mine of the Lake Superior Iron Company. On the right is a V-shaped trough, made by the junction of a diorite mass and a dike. The hard ore is between these and below the Ishpeming quartzite. On the left the hard ore again rests on soapstone, which is upon and interstratified with jasper, and is overlain by the Ishpeming quartzite. Scale, 200 feet to the inch.

Fig. 9 is a cross-section of the Barnum mine, showing hard ore resting either upon folded soapstone or upon jasper, and overlain by soapstone. At the right of the figure is seen a layer of ore between two soapstone dikes. Scale, 200 feet to the inch.

Fig. 10 is a longitudinal section of the Queen, Prince of Wales, and South Buffalo mines, showing the soft ore resting upon an impervious foot-wall of Siamo slate, and grading upward into jasper. Scale, 200 feet to the inch.

Fig. 11 is a cross-section of same mines, showing how the slate is folded into two troughs, which are shown by the longitudinal section (Fig. 3) to have a western pitch. Scale, 200 feet to the inch.

FIG. 8.

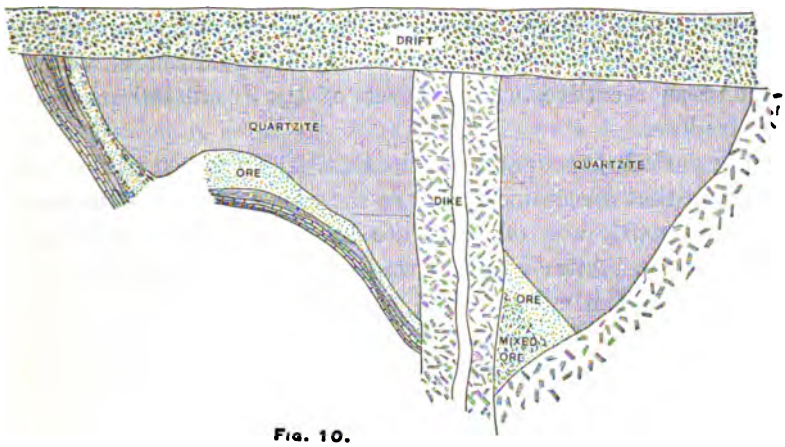


FIG. 10.

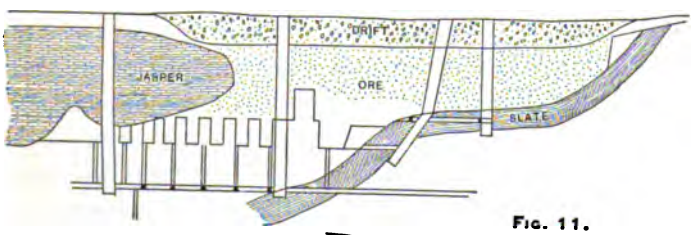


FIG. 11.

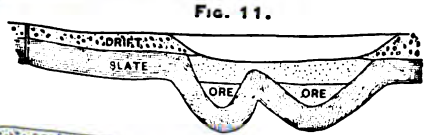
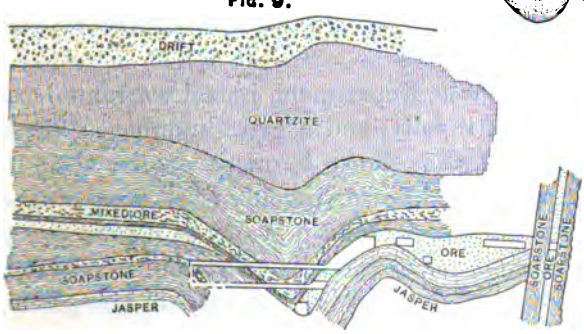


FIG. 9.



The first definite knowledge of the existence of iron-ore in this region dates from the year 1844, when the U. S. government surveyor reported its discovery in running the

township line south from Teal Lake, near Negaunee. Previous to this time there had been rumors of its existence as well as that of other metals. Even as early as 1636, in Lagarde's book of travels, published in Paris, there is a mention of iron; but there is now no generally known account of any attempt to develop the mineral resources of the Lake Superior district during the occupancy of the country by the French, when so much was recorded of the travels of the Jesuit fathers and of the traders.

The period of occupation by the British also passed without any work in that direction, except so far as copper was concerned, and this work was entered into by a company of prominent Englishmen. After a year or more of search for mineral veins from which the "float" had originally come, the enterprise, started by Mr. Henry, was abandoned. This Henry, who was for several years a trader on these shores, gave his opinion that mining of ore in this region would not pay until colonies had been established from which men and supplies could be drawn for the work.

It is possible that the fur-traders were anxious to prevent settlements of any kind; and this is sufficient reason why discoveries of iron-ore such as are reported to have been made were kept a secret. Added to this was the superstitious dread entertained by the Indians of revealing the hidden treasures of the earth. In the early part of this century no exploring was done along the shores of the lake, the U. S. government recognizing the Indian rights up to 1830, when Governor Cass and others made a treaty with the chiefs for this territory east of the Mississippi.

Previous to 1846 the War Department issued warrants leasing mineral lands at a royalty, and a great number of such warrants were taken out, as is shown by maps published up to that date. So far as is now known, the only work done on the Marquette range under these permits previous to the discovery of iron was the sinking of shafts on a small vein of silver-lead, in what is now Presque Isle Park in the city of Marquette. President Martin Van Buren was interested in this enterprise.

When Michigan became a State in 1837 and Dr. Houghton was appointed State geologist, the systematic exploration of the Upper Peninsula was begun. On the failure of the State to raise

enough money to carry on the work in detail, he persuaded the authorities at Washington to combine a geological survey with that of the township- and subdivision-lines; and it was a result of this work that iron-ore was first discovered in 1844 as above stated. The next year a company was formed in Jackson, Michigan, to explore for minerals along the shore of Lake Superior. While at Sault Ste. Marie preparing to embark for Keweenaw Point, whither the exciting finds of copper were drawing men in great numbers, the explorers learned of this iron-ore, and a mining location was afterwards made at what is now the Jackson mine at Negaunee; and upon their return next winter, the excellent quality of the ore was proved by smelting a small quantity in a blacksmith's forge.

The first attempts at mining in this district were made to supply ore for the Catalan forges erected near the mines and also in Marquette, the expense of shipment and the portage at Sault Ste. Marie preventing any but trial-lots of ore being sent away. Considerable iron was made at these forges, but always at a loss to those concerned.

Writing of Marquette in 1851, Mr. Peter White of Marquette says:

"It consisted of a few houses, a stumpy road winding along the lake shore; a forge which burned up after impoverishing its first owners; a trail westward, just passable for wagons, leading to another forge (still more unfortunate in that it did not burn) and to the developed iron-hills beyond, with two or three hundred people uncertain of the future;—they had fallen into the march of the century and were building better than they knew."

In 1850 five tons of ore were sent to A. L. Crawford of Newcastle, Pa. This was the first shipment of Lake Superior iron-ore to Lake Erie ports. In 1852 seventy tons of Jackson ore were sent to Sharon, Pa.

The locks at Sault Ste. Marie were completed in June, 1855, and at the same time a plank road was constructed from Marquette to the mines, and regular shipments were begun. In the following year the construction of a railway was begun, and it was completed in 1857.

This railway was the nucleus of the present Duluth, South Shore and Atlantic railway-system; and extensions to Champion and Republic followed the discovery of these mines, the extension to L'Anse being built in 1872. The Marquette and

Western, now part of the same system, was built in 1884 to connect with the Detroit, Mackinac and Marquette railway. A consolidation of this system was effected in 1887, under the name of the Duluth, South Shore and Atlantic railway.

The Peninsula division of the Chicago and North Western railway was completed from Escanaba to Negaunee in 1864. In 1887 the Chicago, Milwaukee and St. Paul railway was built as far as Republic and Champion; and in 1891 the Iron Range and Huron Bay railway was started from Champion to Lake Superior, but was never operated.

The Lake Superior and Ishpeming railway was built in 1896 by the Cleveland Cliffs Iron Co. and the Pittsburg, and Lake Angeline Iron Co. with docks at Presque Isle, just north of Marquette. Table I. shows the capacity of the docks operated by the above-named companies. It is taken from Mr. John Birkinbine's *Report of the Production of Iron Ore in the United States* for 1895.

TABLE I.—*Capacity of Ore-Docks (From Mr. John Birkinbine's Report).*

	Height.	Length.	Number of Pockets.	Capacity.	Number of Railway Tracks.	Length of Haul from Center of Marquette Range.
	Feet.	Feet.		Tons.		Miles.
Marquette, D., S. S. & A. Ry., No. 1	45.	1715	270	28,000	3	14
3	44.	1260	213	14,000	4	
4	47.6	1280	200	30,000	4	
Escanaba, C. & N. W. Ry., No. 1	46.	1104	184	24,104	2	52
2	39.	1152	192	20,928	2	
3	39.	1356	226	30,284	2	
4	46.	1500	250	32,750	2	
5	51.10	1392	232	43,152	2	
Gladstone, M., St. P. & S. Ste M. Ry	47.	768	120	17,000	5	.....
Presque Isle, L. S. & I. Ry	54.	1250	200	35,000	6	17

Table II. is a statement of the dates of first shipment, total tonnage, etc., of the mines on this range. Besides the names here given there have been a great number of explorations, from some of which small shipments were made.

The figures in Table II. were mostly taken from the Reports of Commissioners of Mineral Statistics of this State, the figures for 1896 having been kindly given to me by Mr. George A.

TABLE II.—*Shipments of the Principal Marquette Range Mines.*

A. ACTIVE MINES.

Name of Mine.	First Shipment. Date.	First Shipment. Tons.	1896 Shipment.	Total Tons.	Character of Ore.	Approximate Depths.	Remarks.
Cambria	1876	6,329	95,086	848,538	S. R. H.	630	Leased from Teal Lake I. Co.
Champion	1868	6,255	113,375	3,082,715	Sp. & Mg.	1300	
Cleveland	1854	3,000	266,872	5,877,442	Sp. & S. R. H.	703	Cleveland Cliffs Iron Co.
Foster	1868	6,000	See Iron Cliffs.		S. R. H.	450	" " " " See Iron Cliffs.
Jackson	Not recorded.		80,710	3,461,685	Sp. & S. R. H.	400	The first iron-mine on Lake Superior.
Lake Superior	1858	4,658	459,575	7,194,389	" "	920	
Lillie	1875	144	107,326	591,557	S R. H.	898	Leased from Teal Lake Iron Co., former "Bessemer."
Mesabi Friend	1873	1,432	10,540	16,043	Siliceous.	.....	Carr and Gribben.
Negaunee	1887	5,259	175,393	823,672	S. R. H.	800	
Pittsburgh and Lake Angeline I. Co.	1864	19,500	342,351	3,694,811	"	500	
Queen	1886	10,860	325,571	2,469,396	"	450	Includes Buffalo, South Buffalo, Prince of Wales, Blue.
Palmer	1895	3,433	1,041	4,474	Siliceous.	.....	
Richmond	1896	1,088	1,088	1,088	"	.....	
Republic	1872	11,025	127,360	4,378,386	Sp. & Mg.	1300	
Salisbury	1872	545	See Iron Cliffs.		S. R. H.	750	Cleveland Cliffs Iron Co. See Iron Cliffs.
Winthrop	1870	3,469	149,437	1,974,350	"	700	Includes Mitchell.
Iron Cliffs Co	.....	.....	245,588	3,606,306	.....	.....	

B. SUSPENDED MINES.

Name of Mine.	First Shipment. Date.	First Shipment. Tons.	Date of Suspension.	Total Tons.	Character of Ore.	Approximate Depths.	Remarks.
Chesire	1872	13,415	1894	217,089	S. R. H.	300	Former Smith, Swanzy included
Cliffs Shaft	1868	14,386	1893	.....	Specular.	600	Including Barnum. See Iron Cliffs Co.
Hartford	1889	566	1895	14,480	S. R. H.	450	
Humboldt	1865	4,782	1895	723,961	Sp. & Mg.	550	Former Washington.
Imperial	1882	1,766	1892	115,076	Limonite.	100	Former Wetmore.
Lucy	1870	4,866	1893	494,280	S. R. H.	400	Former McComber.
Michigamme	1872	141	1891	880,362	Sp. & Mg.	1100	
Riverside	1888	5,622	1890	16,160	" "	200	
Volunteer	1871	4,171	1894	1,073,487	Specular.	500	Former Pittsburgh and Lake Superior.
Star West	1873	1,091	1895	181,004	Soft red Hem	300	Former Wheat or Home.
Dexter	1883	4,878	1896	117,385	" " "	500	And Dey.

EXPLANATIONS: Mg. = Magnetic. Sp. = Specular. S. R H. = Soft red hematite.
Some all-rail shipments to local furnaces are not included in the above table.

TABLE II.—*Concluded.*

C. ABANDONED MINES.

Name of Mine.	First Shipment.		Date of Last Shipment.	Total Tons.	Character of Ore.	Approximate Depths	Remarks.
	Date.	Tons.					
American	1880	1,197	1892	112,933	Specular.	500	Sterling.
Albion	1873	1,186	1879	4,592	"	.....	
Allen	1872	8,707	1874	9,347	S. R. H.	.....	
Ames	1893	1,103	1894	6,298	"	.....	
Bay State	1872	7,633	1882	16,637	"	.....	Indiana.
Beaufort	1882	5,532	1887	90,217	"	400	
Bessie	1891	847	.....	847	Limonite.	100	
Boston	1880	6,478	1884	62,544	Specular.	400	
Chicago	1879	949	1883	9,012	S. R. H.	.....	
Columbia	1873	21,271	1883	94,813	Sp. & Mg.	300	Kloman.
Davis	1884	4,098	1895	121,151	S. R. H.	450	Wheeling and Grand Rapids.
Del & Lackawanna	1886	8,823	1888	17,780	"	250	Sam Mitchell.
Detroit	1882	5,402	1890	140,841	"	600	
East Champion	1873	12,701	1889	76,002	Limonite.	400	Keystone.
Erie	1876	1,058	1883	8,136	Mag.	100	
East New York	1888	13,694	1893	166,248	S. R. H.	500	
Fitch	1890	16,550	1895	31,817	Specular.	150	
Gibson	1885	1,515	1887	16,857	Limonite.	100	
Goodrich	1873	3,25-	1882	49,754	Specular.	.....	
Iron Mountain	1865	898	1875	18,341	Siliceous.	.....	
Manganese	1871	4,787	1879	27,582	Manganese.	.....	
Milwaukee	1879	941	1890	375,451	S. R. H.	400	
New York Hematite	1870	1,809	1882	87,587	"	.....	Grand Central.
National	1878	4,191	1884	150,216	Specular.	400	
New England	1866	3,150	1873	110,506	"	300	
New York	1864	8,000	1894	1,113,102	"	400	York.
North Champion	1887	886	1890	30,574	Limonite.	300	Hortense.
Norwood	1887	2,000	1888	5,753	"	.....	
Platt	1892	2,676	1896	73,844	S. R. H.	200	
Phœnix	1881	10,986	1887	59,114	Limonite.	100	Dalliba.
Pendill	1878	3,386	1884	45,993	S. R. H.	200	
Pascoe	1882	18,880	1886	59,8-6	Limonite.	150	
Rolling Mill	1871	236	1888	234,625	S. R. H.	.....	
Saginaw	1872	18,503	1891	451,424	Specular.	400	
Spurr	1873	31,933	1886	164,244	Sp. & Mg.	400	
St. Lawrence	1882	9,992	1887	23,395	Limonite.	.....	Nonpareil.
Samson	1866	2,843	1892	267,805	Sp & Mg.	500	Edwards or Argyle.
Taylor	1880	1,110	1883	32,970	S. R. H.	.....	
Titan	1882	1,778	1888	90,371	"	400	
Webster	1882	4,443	1889	14,108	"	.....	
West Republic	1881	7,354	1889	133,077	Mg. & Sp.	600	

EXPLANATIONS: Mg. = Magnetic. Sp. = Specular. S. R. H. = Soft red hematite.
Some all-rail shipments to local furnaces are not included in the above table.

Newett, from the advance sheets of his report. From this table it will be seen that the names of some of the first companies still appear as the largest shippers—a good indication of the permanence of their deposits.

The first stimulus to the ore-trade was the Civil War. In fact, there had been no returns to any of the capital invested in the district previous to 1862, when the Jackson Co. declared its first dividend. From this time the yearly shipments increased from 124,169 tons to 1,162,458 tons in 1873—after

which they gained slowly until 1890 when they reached the highest point, namely, 2,993,664 tons.

During the comparatively high prices that obtained until 1890, an enormous amount of exploring was done on the range; but since the opening of the Republic mine in 1872, there have been few profitable undertakings in new territory. Among these few are the Queen group and the Negaunee. The vicinity of the Republic and the Champion mines, with their pure, partly magnetic ores, was favorite ground; but none of the properties opened up near those mines are on the present shipping-lists.

It was not until 1870 that the soft hematite ores of the district were considered to have any value; all the previous work having been done in the hard ores. But in the next few years a large number of pits were opened at the east end of the range in this grade of ore. After the discovery of the Menominee and later of the Gogebic, Vermilion and Mesabi ranges, the number of explorers on this Marquette range gradually diminished. Beginning in 1880, a number of pits in limonite ore were opened near Champion and also near Michigamme. This ore, which was apparently quite shallow and in no case very clean or regular, could be mined with profit only while prices were higher than usual. The last, and the most extensive, mine in this kind of ore, the Imperial, was shut down in 1892.

But a short time after the abandonment of Catalan forges for smelting the ore, the building of charcoal blast-furnaces began. The first was the Pioneer, erected at Negaunee in 1858. This furnace went out of blast in 1893 and the owners, the Cleveland Cliffs Iron Co., have built a furnace at Gladstone, having a capacity of about 125 tons a day. At least 20 of these furnaces were built in the district, and as a usual thing they began by producing less than 10 tons of pig-iron per stack per day. Nearly all of them have been dismantled, and all are now out of blast; though the Excelsior at Ishpeming is expected to start again shortly. Beginning with war-times, and while the supply of hard-wood was abundant, they were very successful. One of these furnaces was intended to burn peat, and two of them anthracite; but they were afterwards changed, for the use of charcoal.

This business did much to stimulate the settlement of the

district, by clearing away the forests, whereby many finds of ore were made, as well as by the direct profits of the undertakings themselves.

The methods employed in the early stages of development of this range, half a century ago, were primitive indeed. For many years—in fact, as late as 1880—most of the ores at Ishpeming and Negaunee were worked in open pits. Tunnels were put in for the railway-tracks, wherever practicable; and after the sand and gravel had been removed, and the ore blasted down, it was loaded directly into the cars. Where the ore had to be raised, use was made of derricks and horse-whims, or else of carts on inclined roads. Tram-cars were first used about 1880; before this time wheelbarrows and trucks were employed.

Hand-drills and hammers were the only means of putting down holes in the hard ore or rock; power-drills driven by steam or compressed air were not introduced until 1878. Nitroglycerin largely took the place of powder in the mines about 1869. The development of these irregular bodies of ore was aided to a marked extent by the introduction of diamond-drills, by the use of which a core could be taken, proving the ground in some places to the depth of 1000 feet. With them it was possible to locate an ore-deposit situated under a swamp, or, by drilling through the ice of a lake, to determine the presence of enough merchantable ore under the bed of it to warrant a large expenditure of money—as was the case with Lake Angeline.

An interesting estimate of the cost of mining in Lake Superior and a comparison with similar cost in Sweden, from Major Brooks's State Geological Report of 1873, pages 255 and 256, is republished in Table III.

The system of stoping hard ores, and leaving pillars enough to support the hanging-wall, was the one generally adopted. Many of the mines opened under this plan have been since filled, and the pillars have been removed.

The soft ores were usually worked with square sets of timber, as in the mines of the West. This system paid when the price of that kind of ore was as high as in 1890; but it has been largely abandoned for the caving-system, variations of which, to suit the different hardness of the ore or surrounding material, make up the plan now usually adopted.

TABLE III.—*Comparative Costs of Mining, Taken from T. B. Brooks's State Report*, 1873.

COST OF MINING HARD ORE ON LAKE SUPERIOR.			COST OF MINING HARD ORE IN PERSBERG MINES, SWEDEN.		
	Per Cent.	Based on Total of $2.64 per Ton.		Per Cent.	Based on Total of $2.20 per Ton.
Dead-work	28.1	$0.742	In the mines	38.66	$0.85
Mining-labor	39.8	1.050	Pumping	3.50	0.07
Materials, tools, etc	11.9	0.313	Hoisting	11.17	0.26
Handling and pumping	15.6	0.413	Dressing	8.12	0.18
Management	4.6	0.122	Picking	5.65	0.12
			Buildings	16.45	0.36
			General expenses	16.45	0.36
Total	100.0	$2.640			
			Total	100.00	$2.20

The use of timber in such large quantities as are required for the mines has depleted the immediately adjacent forests, from which the hard-wood and the best of the white pine had already been cut. Wood is now brought a considerable distance by rail or water. The principal kinds used are the Norway and some white pine, generally of too poor a grade for lumber; these, with a small quantity of hemlock, are used for legs and caps. In size they run from 6 or 8 inches through up to 24 inches, or higher, according to the nature of the ground and the permanence of the work. Tamarack and some spruce is used for ladders, poles, etc., while cedar, being light, is the favorite for lagging or blocking. Coal has almost entirely taken the place of wood for fuel, being brought as a return freight from Lake Erie ports by the fleets of vessels owned by the different mining companies.

No water-power has been used for generating electricity for the mine. At Republic the air-compressors are driven by the falls of the Michigamme river.

Table IV. shows the shipments from the Marquette range from the start, together with the reported selling prices and also vessel freight-rates.

It will be seen that the Marquette range has shipped 46,542,662 tons of iron-ore, or nearly half of the entire Lake Superior product to date.

Since 1890, owing to the rapid development of the large mines on other ranges, and the dull times prevailing in the

TABLE IV.—*Shipments from the Marquette Range.*

	Total Shipments.	Price of Ore at Lake Erie Ports.		Vessel Rates per Ton to Lake Erie Ports.	
		Bessemer.	Non-Bessemer.	Marquette.	Escanaba.
	Tons.	Dollars.	Dollars.	Dollars.	Dollars.
1854	3,000	.....	.....	.....	.....
1855	1,449	.....	.....	.....	.....
1856	86,343	8.00	8.00	3.00	.....
1857	25,646	8.00	8.00	3.00	.....
1858	15,876	6.50	6.50	2.00 to 2.50	.....
1859	68,882	6.00	6.00	2.00 " 2.50	.....
1860	114,401	5.25	5.50	2.00 " 2.50	.....
1861	49,909	5.25	5.00	2.00 " 3.00	.....
1862	124,169	5.25	5.37	2.25 " 4.50	.....
1863	208,055	7.50	7.50	3.00 " 4.00	.....
1864	248,127	8.50	8.50	3.00 " 5.00	.....
1865	286,208	7.50	7.50	2.50 " 5.00	.....
1866	278,796	9.50	9.50 to 14.00	2.75 " 6.50	2.50 to 5.75
1867	473,567	10.50	8.00 " 11.50	2.00 " 4.00	1.50 " 3.50
1868	491,449	8.25	8.25	2.25 " 3.25	1.60 " 2.60
1869	617,444	8.25	9.50	2.75 " 4.10	1.50 " 2.50
1870	830,940	8.50	8.50 to 9.50	2.50 " 3.25	1.65 " 2.50
1871	779,607	8.00	8.00	2.25 " 4.00	1.50 " 2.50
1872	900,901	9.00	7.50	2.85 " 6.60	2.00 " 5.25
1873	1,162,458	12.00	9.00	3.25 " 4.00	2.30 " 3.00
1874	919,557	9.00	7.00	1.60 " 2.50	1.30 " 1.40
1875	891,257	7.00	5.50	1.30 " 1.50	1.10 " 1.30
1876	992,764	6.75	4.50	1.25 " 2.20	0.70 " 1.45
1877	1,010,494	6.50	4.25	1.25 " 2.00	0.65 " 1.50
1878	1,033,082	5.50	4.25	1.00 " 1.50	0.60 " 1.15
1879	1,180,019	6.25	4.75	1.25 " 3.00	0.70 " 2.10
1880	1,384,010	9.25	8.00	2.00 " 2.75	1.50 " 2.00
1881	1,579,834	9.00	7.00	2.00 " 2.30	1.00 " 1.60
1882	1,829,394	9.00	6.25	1.25 " 2.00	0.90 " 1.40
1883	1,305,425	6.25	5.00	1.30 " 1.75	0.90 " 1.50
1884	1,559,912	5.76	4.50	1.00	0.65
1885	1,430,863	5.50	4.25	1.40	1.00
1886	1,627,383	5.50	4.75	1.75	1.35
1887	1,851,717	7.25	5.25	2.15	1.75
1888	1,918,672	5.50	4.75	1.10 to 1.15	0.90 to 1.45
1889	2,684,817	5.50	4.50	1.10 " 1.25	0.90 " 1.25
1890	2,993,664	6.75	5.75	1.00 " 1.25	0.85 " 1.05
1891	2,511,395	6.00	4.75	0.70 " 1.40	0.55 " 1.25
1892	2,680,194	5.50	4.85	0.80 " 1.15	0.65 " 1.00
1893	1,832,750	4.25	3.00 to 3.50	0.50 " 0.95	0.45 " 0.80
1894	2,059,241	2.75	2.15	0.55	0.40
1895	2,095,166	2.75 to 3.50	2.15 to 2.30	0.60	0.45
1896	2,473,051	4.00	2.45 " 2.85	0.66	0.52
1897	*	2.65	2.00 " 2.60	0.50	0.45
	46,542,662				

iron-trade since 1892, the product from the Marquette range has not increased, many of the smaller mines and some of the larger ones having suspended operations or abandoned works altogether.

The causes above stated have forced the mining companies to adopt a policy of strict economy, in methods of mining especially, under which the cost has been materially reduced. This reduction is partly owing to the fact that the mines had been previously opened up far ahead of their present needs, and had been equipped with modern machinery.

* It is thought that the shipments of 1897 will approximate those of 1896.

Ishpeming and Negaunee remain the center of mining on the range; in fact, except Republic and Champion, nearly all the ore shipped is from their city limits.

The iron business has been the chief industry of the district, though attempts have been constantly made to develop its other resources. The beds of brownstone at Marquette have been worked for over 25 years. Among the ventures in which large sums have been spent there may be mentioned the slate quarries at L'Anse, which were worked intermittently for 20 years, but proved too expensive to operate without railway facilities.

The Ropes gold-mine, opened in 1881, is still working, and has produced something like $600,000 worth of gold and $50,000 worth of silver, though it has not been a source of profit to its owners. The shaft is down to the 850-foot level, and the vein holds out about the same as near the surface. The average yield is from $2 to $3 per ton; the ore-body is 550 feet long, and the formation has a width of from 30 to 50 feet.

Work at the Michigan gold-mine, in which such rich pockets were found, was suspended in 1893. It may well be thought that a larger amount of capital may some day be rewarded in this field, which has promised so much.

Many explorations have been made on small veins of silver-, lead- and copper-ore, but not enough has been found to warrant further work in them.

There is fine granite in large quantities favorably located near Lake Superior; and immense hills of *verd' antique* marble, near the railway, awaiting capital to work them.

The development of the ores of the range has been greatly assisted by the well-known geologists who have been connected with the United States and the State Survey.

Dr. Houghton has already been mentioned. He was followed by Foster and Whitney, who published in 1850 the results of their explorations.

The work has been ably carried forward in different lines by such men as Brooks and Pumpelly, Charles E. Wright, Dr. Wadsworth, Dr. Hubbard and Dr. Van Hise, who have all published valuable reports at different stages of its development.

Some Dike Features of the Gogebic Iron-Range.

BY C. M. BOSS, BESSEMER, MICH.

(Lake Superior Meeting, July, 1897.)

THROUGHOUT the Lake Superior Basin each of the great iron-ranges, from which vast quantities of iron-ore have been, and are now being mined, presents characteristics differing from each and all of the others.

One of the most peculiar of these varying features is found in the dikes of the Gogebic range. From an economical standpoint this feature of the dikes has been studied by our practical mining men more than any other geological phenomena in connection with the deposits of merchantable iron-ore. In the greater portion of our mines these dikes appear to form the riffles in the sluice of the ore formation, against which the particles of iron oxide are deposited in the process of concentration out of the quartzose matrix which is termed the iron-bearing formation.

In composition these dikes are apparently of a dioritic formation, more or less altered and decomposed, comparatively soft, and sometimes almost steatitic. While frequently showing something of a cleavage in the direction of their pitch, they are devoid of parallel laminæ, and when broken assume the form of irregular angular fragments. They are generally of an ashy grey color, but sometimes at great depth are found to be quite dark, almost black, and of a very hard massive nature, showing radial or fibrous crystallization.

Practically all of the dikes thus far discovered in the iron formation are approximately at right-angles with the dip of the formation, and the greater number of them have an average easterly dip of from fifteen to eighteen degrees. Sometimes, however, they are folded in such a manner as to form long synclinal basins; but while their western or initial extremity is fre-

quently found at or near the surface, their eastern extension, so far as they have been followed, invariably shows a downward tendency. There are several instances in which they have been found to be nearly vertical, but wherever the slightest inclination is shown it is always to the east.

In the early history of mining operations on the Gogebic range it was thought by many that a large dike, that is, a very thick one, was the bottom of all things so far as iron-ore was concerned, and that nothing of any value would be found by sinking through them. But as the first dikes were found at or near their western extremity, and all lying *en échelon*, with an easterly pitch, the reasonable deduction should have been that if the ore-body overlying one dike was continuous, it would at some depth be found under the next dike to the eastward.

The inquisitive nature of the American miner soon required some evidence more tangible than opinion or theory as to what would be found under the dikes. Shafts were continued through them; and in a majority of cases these efforts were rewarded by finding another lens of ore lying upon a similar dike, and in some instances succeeding ore-lenses and dikes were found to the number of three or four. In most cases, however, ferruginous quartzite of varying thickness was found immediately underlying each dike, forming a cap to the succeeding deposit of ore.

In 1889 the Metropolitan Iron and Land Co. caused a diamond-drill hole to be bored near the west end of the Norrie mine. This drill-hole, started at a point 982 feet north of the surface-line of the foot-wall, and sunk vertical, entered the foot-wall at a depth of 1500 feet, thus indicating the average dip of the foot-wall to be 56 deg. 45 min. The records of this drill-hole show that nine successive dikes are to be encountered in mining before that vertical depth is attained. The existence of large bodies of ore above and below the upper three dikes has already been proved by actual mining operations; and the evidence of the drill-hole is to the effect that large deposits of ore will be found lying on the foot-wall both above and below the lower two dikes.

From the present development of the iron-formation on this range, I can see no valid reason why a succession of dikes and

deposits of ore may not reasonably be expected to be found to a depth at least equaling the limits of economical mining.

While the larger dikes so far discovered are quite regular, and in extent exceed the limit of mining operations, there are numerous small dikes, sometimes apparently independent, sometimes splits or offshoots from others. In a few instances dikes having the appearance of great strength have been known to end abruptly, and occasionally large isolated bunches of dike-material are found imbedded in the ore-body.

The geological theory of dikes is that they originate at some point below all sedimentary rocks, and that they have been injected through weak places and crevices in the overlying strata; therefore it must be admitted that these particular dikes must at some point cut through the foot-wall. In many instances this fact has been demonstrated; but several cases have come under my notice where, temporarily at least, different strata of the iron-formation have successively resisted the lateral extension of a dike, causing it to deflect, or fold upon itself, or to terminate abruptly. It is possible that an abrupt termination may be a faulting, and the continuation may be discovered in future mining operations.

It may be appropriate at this time to give a short description of the characteristics and peculiarities of the principal dikes which have been exposed in the prosecution of mining operations along this range.

The most westerly dike of which we have any definite data has its initial at the surface in the S.E. ¼ of the S.E. ¼ of Sec. 10, T. 46 N. of R. 1 E., Wis., about 1200 feet from the east line of the section. Its location is again fixed by a diamond-drill hole at a point 600 feet east of its initial, where it is overlaid by a fine body of high-grade ore known as the Shores mine; and 900 feet further east it is touched by the No. 6 shaft of the Iron Belt mine at a depth of about 450 feet from the surface. The thickness of this dike is not known, but its pitch to the east is quite regular.

About 2500 feet further east, near the center of Sec. 11, the feathered apex of a dike is shown in the workings tributary to No. 5 shaft of the Iron Belt mine. In the second level of this shaft, which is 150 feet below the surface, a drift has followed

the dike eastward for 280 feet; and in one portion of this drift is seen the peculiar phenomenon of the dike abutting abruptly against a hanging-wall of black slate. In the cross-cut from the shaft at this level the dike is about 20 feet in thickness, but in the eastern end of the drift it has narrowed to 5 feet. In the first level, which is 57 feet above the second, it is 10 feet thick in the cross-cut and in the workings to the west, but diminishes to a sharp apex about 10 feet above the level. It does not appear on this level east of the cross-cut. From the general trend of what can be seen of this dike, I should judge that its western initial was not more than 800 feet west of No. 5 shaft.

From No. 5 shaft the ground rises rapidly to the east, and on the N.E. ¼ of Sec. 11 are located the four shafts of the older portion of the Iron Belt mine. In this rising ground is the commencement of another dike on which rested the ore-body penetrated by these shafts. This dike is quite a strong one, but towards the east side of the property is somewhat broken and faulted, with a corresponding irregularity in the volume of ore. Further east, in the Atlantic mine, the natural conditions of both dike and ore obtain, and at 700 feet in depth the former is seen at its uniform dip and pitch. The Atlantic shaft has been sunk to a point 75 feet below this dike, but no further lens of ore has been encountered.

In the 2 miles between the Atlantic mine and the Pence mine there has been considerable exploring for iron-ore, but nothing of moment has been discovered. At the Pence mine a large lens of high-grade ore was found lying upon a dike of extreme regularity. The ore of this lens has all been removed from the Pence and Father Hennepin lands, and practically from the Montreal property, and is now being wrought by the Montreal Mining Co. on the "Sec. 33" property, farther east. This dike is about 40 feet thick, and the trough formed by its intersection with the foot-wall is so regular that the location of it on the "Sec. 33" property, from data obtained by personal surveys in the Pence and Father Hennepin mines, nearly three-quarters of a mile distant, was found by development to be correct within 2 feet.

In the properties between the Montreal mine and the Ash-

land mine at Ironwood, including the Superior, Cary, Windsor and Germania mines, some small dikes have been noticed, but nothing of sufficient importance to affect the ore-formation materially.

In the workings of the Ashland, Norrie, East Norrie, Aurora, Pabst and Newport mines may be seen the greatest development of the dike-phenomena. The operations on these properties are practically continuous for 2½ miles, and have been carried to greater depths than on any other portion of the Gogebic range. The greatest number of dikes encountered on any one property have been traced in the Ashland mine, and located with reasonable accuracy. The first one to enter the western limit of the property is first seen in a shaft of the Minnewawa mine on the west bank of the Montreal river, and is cut by the No. 1 shaft of the Ashland mine at a depth of 200 feet from the surface. This dike is traced in a generally regular manner through the workings of the various shafts, to a point between the 9th and 10th levels in No. 8 shaft, whence it emerges from the property below the workings of the Norrie mine. There are three other dikes below this one, the upper of which is first seen immediately below the 7th level in No. 3 shaft, and passing eastward disappears below the 10th level a short distance east of No. 7 shaft. The second one is found at the 9th level in No. 3 shaft, and just below the 10th level in No. 4 shaft, whence it rises to a horizontal position, and gradually, thinning out, disappears before reaching No. 5 shaft. From such data as could be obtained in the drifts west of No. 3 shaft a plausible theory may be advanced that these two last-mentioned dikes may be offshoots or splits from the one first above described. The third dike was cut in shafts Nos. 3 and 4, below the 10th level.

A large dike near the surface in No. 2 shaft passes through Nos. 3 and 4 to just below the 7th level in No. 5 shaft, whence it runs horizontally through the mine, and is cut by the No. 2 shaft of the Norrie mine at the 8th level, just east of which it dips below present operations. This is the third dike thus far encountered in the Norrie mine.

A dike having its initial just east of No. 4 shaft of the Ashland mine passes eastward through the property, cutting

the most easterly, or No. 8 shaft, immediately below the 3d level. This forms the second large dike of the Norrie mine, and extends from the 4th level of the No. 1 shaft to the 10th level in No. 6 shaft, whence it passes below present workings.

The greatest longitudinal development of any dike yet located on the range is found in one having its initial about midway between shafts Nos. 7 and 8 of the Ashland mine, entering the Norrie mine at the 3d level and gently descending eastward to the 7th level in No. 6 shaft, and to the 8th level in No. 1 shaft of the East Norrie, thence generally horizontal to the No. 1 shaft of the Aurora mine, whence it inclines to below the 10th level in No. 3 shaft and passes beyond observation. This is the first or main dike of the Norrie and East Norrie mines, and the second dike of the Aurora. It is the base of the largest deposit of iron-ore yet developed on this range. Its total length thus far exposed is 6200 feet, and its general characteristics are such that it may be expected to continue eastward for a much greater distance.

In the East Norrie mine a crossing dike is found which passes at a very high angle from the surface at No. 1 shaft to the 10th level and downward at No. 2 shaft, cutting the ore-formation transversely. It is from 15 to 20 feet in thickness, and so far as can be observed, is cut by the main dike at the 8th level, thus proving it to be of previous origin.

One of the dikes first discovered on the range is the one underlying the original open pit of the Aurora mine, and starts from surface just east of the No. 1 shaft. This dike inclines to the east quite rapidly, and is found at the 8th level in the No. 6 shaft. It continues on the Pabst property, where it forms the second dike, which is cut in the most westerly or "C" shaft. Under the center of the Pabst mine this dike rises, forming the first synclinal basin that we have noticed. The dike above this is first seen on the Pabst property under what is known as the north vein. It is parallel with the lower one and forms a similar basin. These two dikes continue to rise to the east through the Newport mine; but it is not known whether they come to the surface, or take a downward pitch again, to the east of the mine-workings. The lower dike appears to cut the foot-wall, but the upper one does not; and it is presumably

faulted from the same causes that produced the division of the ore-body forming what is termed the north and south veins. In the Newport mine are two vertical dikes crossing the formation at right-angles. One is about 100 feet east of the "C" shaft, and is 10 feet thick; the other is 200 feet further east and about 20 feet thick. It is not determined whether these are cut by the horizontal dikes or not.

Passing eastward from the Newport mine the development of the various properties has not passed the exploratory stage, and no dikes worthy of note have been located until reaching the east half of the old Puritan mine on Sec. 17, T. 47 N., R. 46 W. Here a dike is found which forms the base of the ore-deposits which have been worked on the Puritan, Ironton, and Federal mines on Sec. 17, and the Jack Pot on Sec. 16, at which point it descends below the present range of mining operations. This is a strong dike and has never been penetrated sufficiently to determine the character of the formation beneath.

The next located dike is the one beginning under the original open pit of the Colby mine. The first mining operations on this property consisted of open-pit work on what were then known as the north and south veins. These two openings were about 200 feet apart, and the ore in each was bottomed on dikes lying perpendicular to the dip of the formation. The north vein has proved to be but 500 or 600 feet in length, and in going down is merged into the south vein. At this point the south vein dike dropped very suddenly and united with the other, forming the main dike on which lies the ore-body of the Tilden mine. The workings of this mine have followed this dike for nearly half-a-mile to the east, where it drops below the present bottom levels. The No. 7 shaft of the Tilden penetrated this dike at a depth of 625 feet from surface, finding ore underneath. At this point the dike was but 12 feet in thickness, and at 120 feet below this the shaft is bottomed in another. From the conditions observable here it seems most probable that the 12-foot dike is but an offshoot from the lower one.

In the Palms mine, immediately east of the Tilden, are found one or two small dikes which do not appear to greatly affect

the formation; but in the Anvil mine adjoining the Palms a large dike comes in, rising towards the east; and at a depth of about 300 feet from the surface, continues eastward in an undulating but generally horizontal plane for nearly three-quarters of a mile, where, in the eastern end of the Eureka mine it suddenly pitches down again. No ore was found under this dike in the Anvil mine; but in the Eureka a good body was found after passing through 150 feet of cap rock.

The hitherto great regularity of the iron formation is broken at the valley of the Black river near the Eureka mine, and it is widely spread out to the eastward. A large amount of exploring has been done on the general trend of the mining range, but ore in merchantable quantity has been found only at the Mikado mine on Sec. 18; to the north and east of Sunday Lake, on Secs. 9 and 10 in T. 47 N., R. 45 W.; and at the Comet mine on Sec. 11 of the same township. The shaft at the Mikado mine is sunk for a long distance in a dike, but sufficient work has not been done to determine its dip or pitch.

A large dike is found to the west of the shaft in the workings of the Sparta mine, north of Sunday Lake. It is nearly at right-angles with the trend of the formation, and almost vertical, having only a slight dip to the east. This dike is 62 feet in thickness, and has a deposit of ore on the east side, ferruginous quartzite appearing on the west.

The Brotherton and Sunday Lake mines, just east of the Sparta, have been wrought to a depth of 500 feet, but have as yet encountered no dikes. A small one was seen in the bottom of one of the Comet shafts, but aside from this, so far as I am aware, none were ever discovered in the extensive explorations between Sunday Lake and Lake Gogebic.

The Genesis of Certain Auriferous Lodes.

BY JOHN R. DON, OTAGO, NEW ZEALAND.

(Chicago Meeting, February, 1897.)

Introduction by the Secretary.

This paper, under the title of "The Genesis of Auriferous Lodes from a Chemical Point of View, Illustrated by Analyses of Samples Taken from the Chief Auriferous Area of New Zealand, Victoria and Queensland, by John R. Don, D.Sc., M.A., Lecturer on Geology in the University of Otago, N. Z.," was submitted by the author with the frank confession that its length, covering several hundred printed pages of the *Transactions* of the Institute, would preclude its acceptance for publication in full. But the great value of the original work which it records rendered its rejection on that ground highly undesirable; and, after correspondence with Dr. Don, it was agreed that the Secretary should condense the paper, subject to the author's approval, omitting what was not essentially connected with the original work reported. In the discharge of this laborious and difficu't duty, the Secretary's chief trouble has been his regret to cut out the acute criticisms and admirable theoretical and historical summaries of Dr. Don on the general subject of the science of ore-deposits. It should be added, that the original paper has been returned to the author, with full permission to publish it through any other medium (due mention being made of the first publication by the Institute of portions thereof), and a cordial expression of the hope that the treatise, as a whole, may be thus published, to the advantage of science. The Secretary begs to add, that many of the portions necessarily omitted from this condensed version would constitute, in his judgment, interesting and valuable separate contributions.

In attempting to condense this paper, it was necessary at the outset to cut out the first three chapters, of which the following brief outline is therefore given.

Chapter I., introductory in character, indicates the importance of chemical analysis in the investigation of the genesis of ore-deposits, emphasizing the value of Prof. Posepny's Institute paper, and especially of the discussion which it aroused, and stating in general terms the theories of "lateral secretion" and "ascension."

Chapter II. describes the scope of the investigation undertaken by the author, the chief purpose of which was to determine which of the above-named theories was favored by the evidence obtained. This evidence consists chiefly (though not wholly) of analyses of the country-rock taken at various distances from the auriferous "reefs," (*a*) on deep mine-levels; (*b*) in the surface or "vadose" region; and also (*c*) of underlying, neighboring or connected crystallines (granite, gneiss, etc.). In addition to these investigations, there is a separate inquiry into the question of the deposition of gold in marine basins, suggested by the fact that most of the gold in the lodes of Australasia occurs in stratified deposits.

Chapter III. discusses the principles and methods followed by the author in the examination of country-rock for the purpose described. After giving his reasons for believing that careful tests of carefully-chosen samples of country-rock taken at varying distances from the ore-deposit might throw light upon the question

whether the gold in the deposit was derived from the adjacent country-rock, he says he sought a method by which one-fourth, or at most one-half grain of gold per ton of 2240 pounds, might be estimated, and describes his trials of several methods, namely :

1. That described by W. Skey,* involving the use of a dilute solution of iodine or bromine, which the author found unsuitable for his purpose.

2. Chlorination, which was likewise discarded, for reasons given.

3. The fire-assay, which, with certain described precautions for the purification of the litharge employed, was finally adopted.

4. The use of alkaline cyanide as a solvent, with subsequent precipitation, was not tried.

In the majority of cases the samples taken weighed 7 or 8 pounds; the samples tested 4.48 pounds; and the assays were made from concentrates of the latter, but sometimes 4.48 pounds was the actual sample for assay; in which case the total quantity was pulverized to pass a No. 60 sieve, and divided into twelve parts of about 2500 grains each, and to each part 3000 grains of purified litharge, 2000 grains of carbonate of soda, and 1000 to 1500 grains of borax were added for the assay, with sufficient argol to reduce about 400 grains of lead. Further precautions as to cupelling and parting are also given.

Very minute gold-prills, instead of being weighed, were measured by means of the microscope; and the author found that the method suggested by Mr. G. A. Goyder, of Adelaide, S. Aust., and published by him in the *Chemical News*, vol. 70, 1891, pages 194 and 202, had many advantages. According to this method the prill to be measured is enclosed in a bead of boracic acid.

The author says he soon found that samples of country-rock containing no sulphides were not likely to contain gold, and that it was consequently more convenient (and quite sufficient, if control-assays of residues were made when desirable) to concentrate the crushed samples of 4.48 pounds, and assay the heavier portions only. After testing several mechanical methods (of which Thoulet's was the most satisfactory), he found that the best of all was careful panning by an experienced operator. *In the case of each mine*, at least two samples of the residues from panning, the concentrates of which had shown no sulphides, *were assayed*. The results confirmed the general experience that, in the deeper (unoxidized) zone, the gold always accompanied sulphides.

The percentage of sulphides in concentrates was determined, after the removal with a magnet of all magnetic particles (magnetite, iron from mortars, etc.), by sampling the residue, and separating sample (generally 5 grammes) by means of Sonstadt's solution of mercuric iodide dissolved in potassic iodide, or of a solution of boro-tungstate of cadmium, which was used with a specific gravity of 3.16 to 3.18. This gave two classes, one of which had more than 3.18 sp. gr., and was reckoned as sulphides, the percentage of which in the whole concentrate was easily calculated. The percentage of sulphur contained as insoluble sulphides in the rock was also determined separately; and the result of these analyses is shown in the various tables and diagrams under the heading "Percentage of Sulphur." In one case (New Chum Railway Mine, see Table I., on page 622), systematic tests of rock for carbon were made. In other localities, this was not done, as there seemed to be no general relation between the percentage of carbon in the country-rock and the distance from the reef.

The succeeding chapters have been condensed from the author's manuscript. It should be added here, that in the original paper, as presented, the tables were

* "On the Application of Iodine and Bromine for the Detection of Gold when in Minute Quantities," *Trans.* N. Z. Inst., ii., 156.

illustrated by numerous sections of mine-workings, showing the precise localities from which the samples, *a*, *b*, *c*, etc., had been taken.

Since the presentation of this paper, Dr. Don's address has been changed to the Waitaki High School, Oamaru, N. Z.

Chapter IV.—Analyses of Deep Country-Rock.*

The districts to which these investigations refer are: (1) in Victoria, those of Bendigo (Lower Silurian), Ballarat and Steiglitz (Lower Silurian), and Walhalla, Gippsland (Upper Silurian); (2) in Queensland, Gympie and Charters Towers; (3) in New Zealand, Macetown, Otago, South Island of New Zealand; Reefton, Nelson, South Island of New Zealand, and Thames, North Island of New Zealand.

Bendigo, Victoria.

The most important auriferous areas of Victoria occur in long and comparatively narrow belts, striking, practically with the enclosing Lower and Upper Silurian rocks, northerly and southerly. Bendigo occupies one of these belts.

For detailed descriptions of this district, reference is made to the monograph of E. J. Dunn, published by the Victorian Mines Department at Melbourne, 1893, and to the papers of Mr. Rickard in Vols. XX. and XXII. of these *Transactions*.

The New Chum Railway Mine.—The first of the mines selected for this inquiry was the New Chum Railway Company's mine, one of the deepest on the New Chum reef, which is one of the three most productive anticlinal axes in Bendigo.† On the 2498-foot level of this mine, a cross-cut extends 265 feet east from the main shaft, and at the end of it a winze was sunk to a further depth of 360 feet; and on driving about 12 feet east at the bottom of this winze, the west leg of a saddle-reef was struck. At the time of my visit, January, 1895, this showed about 4 feet of laminated quartz, yielding 1.5 ounces of gold per ton, and was the deepest pay-ore exposed in Australia up to that time. (A later discovery has been reported from Lansell's "180" mine on the same anticlinal, at a depth of more than 3000 feet.) Samples were taken along the above-mentioned cross-cut from the shaft to the winze (see *a* to *q*, Table

* This chapter contained many descriptions and maps of districts and mines, which have been omitted.

† See Mr. Rickard's two papers, already cited.

I.). At 190 feet from the shaft, a slide or slight fault was crossed, from the sides of which three samples, two of slate (*l*, *m*) and one of sandstone (*k*) were taken. At the bottom of the winze, 2858 feet from the surface, three samples of slate (*s*, *t*, *u*) and one of sandstone (*r*) were taken close to the reef, to show the difference, if any, between the country-rock in close proximity to a reef and that further away. Table I. gives the results of the examination of the samples to determine their gold-contents per ton, and their percentage of sulphur and of carbon, and Diagram 1 shows them graphically.

The South St. Mungo Mine.—The second Bendigo mine selected for investigation was that of the South St. Mungo Co., situated on the New Chum line of reef, but presenting a modification of the saddle-reef type, in which the lode fills a more or less continuous fissure in "center-country," *i.e.*, along an anticline such as is usually occupied by the saddle-reefs. The possible effect of this variation in lode-form and structure upon the distribution of gold in the reef and rock respectively, was the reason of choosing this mine for study. In this case the lode follows a nearly N.—S. anticline, and dips slightly E. It has been worked to 1180 in feet depth, and carries an enormous quartz body, 20 to 60 feet wide, with a remarkably small percentage of sulphides (chiefly pyrite and blende, with traces of galena). At the 1180-foot level I saw in January, 1895, this immense mass of white, splintery quartz. On the 610-foot level a body of quartz 60 feet wide was worked throughout the whole length of the claim, and this level alone is said to have paid over £50,000 in dividends. The cross-cut at the 1180-foot level, which struck the reef 128 feet E. of the shaft, was sampled (see Table II. and Diagram 2). By reason of the steep dip of this lode, the cross-cuts from the shaft to it are comparatively short. But a prospecting cross-cut driven E. 302 feet at the 610-foot level gave an opportunity for getting samples further from the lode (see Table III.).

Tables I., II. and III. show a striking connection between the percentage of sulphur (as sulphides) and the contents of gold in any sample, especially from the slates; the sandstones, particularly at a distance from the reef, sometimes containing large crystals of non-auriferous pyrite. The sandstones generally carry much less sulphide than the slates, and the sulphides

in sandstone are rarely, while those in the slates are usually, auriferous. (See analyses *e*, *h*, *i* and *r* in Table I. and the whole series in Table III.)

A marked inverse relation between auriferous contents and distance from the lode is observed. The zones of country-rock containing appreciable quantities of gold are confined to 50 or 60 feet on each side of the lode.

Samples from the vicinity of a slight fault, even at some distance from a main reef, contain much more gold and sulphides than the adjoining rocks, away from the fault.

Another remarkable fact which seems unaccountable is that the samples of slate from the New Chum Railway Co.'s mine (Table I.) show a striking increase in carbon as the reef is approached. Mr. L. A. Samuels, an experienced mine-manager in Bendigo, has assured me that this is a common phenomenon on the line of another reef in that district.

Finally, it is observable that the mine-samples (of 4.48 pounds) in which sulphides did not occur contained no gold or silver.

Ballarat, Victoria.

This district comprises two narrow belts running almost due N. and S., one passing through Ballarat West, and worked pretty continuously for about 5 miles; the other passing through Ballarat East, and worked for about 8 miles. The two belts differ in vein-formation and in vein-materials.

In the western belt the Upper Silurian shales and sandstones, constituting the country-rock, have been much folded, are now almost invariably steeply tilted, and are traversed by true fissures, striking almost always N. and S., filled with quartz, and receiving numerous "leaders" and "droppers" from the country on both sides, the whole sometimes forming a "stockwork" many feet wide.

In the eastern belt the strata have been bent, as at Bendigo, into sharp synclines and anticlines. But in this case, where fracture has occurred along the anticlines, the result has been, not "saddle-reefs," as at Bendigo, but more or less continuous fissures filled with quartz. The most interesting and characteristic feature of the deposits of Ballarat East, however, is not the occurrence of these continuous lodes, but that of the so-called "indicators." These are certain members of a series of thin-

bedded sandstones and slates, which have been traced along the whole length of the belt, so far as the latter has been explored. The indicators, which have exerted an important influence on the gold-bearing character of the quartz-lodes traversing them, are thin beds of dark carbonaceous shale, abounding in pyrite and arsenopyrite, the sulphides in some places completely replacing the shale of the bed. The indicators are conformable with the bounding slates and sandstones, which stand almost vertical, and they share with the whole series in the effect of numerous dip-faults, or minor dislocations on the dip, locally termed cross-courses. Several of them run parallel within a band some 300 feet wide, and one, known as "The Indicator," has been traced N. and S. nearly 8 miles. (See Fig. 1, copied from a drawing by Mr. T. A. Rickard.)

The quartz-bodies in this indicator-belt are not regular reefs, but veins of varying thickness crossing the indicator at all angles; those which present a nearly horizontal crossing, with a strike parallel to the indicator and a slight dip E., having proved most productive. Away from the indicator, the greater part of the vein-quartz is absolutely barren; but at the intersection with the indicator large masses of gold (often more than 100 ounces in one piece) have been obtained, and the greater part of the gold extracted from this belt has come from those parts of the quartz-veins near some one of the indicators.

Mr. E. Lidgey, of the Victoria Geological Survey, in his *Special Report on the Ballarat East Gold-Field*, gives a list of 15 gold-masses or nuggets found as above described, and varying in weight from 30 to 137 ounces, and in depth from the surface (measured on the indicator) from 160 to 805 feet, the average being 470 feet. In the same report he raises several questions as to the source and distribution of the gold in the indicators, and says that the solution of these problems will require a series of investigations of the country-rock, like those performed by Sandberger in Germany.

The Prince Regent Mine.—Before seeing Mr. Lidgey's report I had taken a set of samples for the purpose suggested, selecting the Prince Regent mine, one of the most productive on the indicator belt, as likely to be most instructive from a genetic standpoint, because it had shown, as the depth of the mine increased, a marked change in the auriferous contents of the

veins crossing the indicators. This mine is about 870 feet deep. Down to the 630-foot level, pay-quartz, sometimes with

FIG. 1.

VERTICAL SECTION OF THE INDICATOR, BALLARAT-EAST.
Showing quartz veins.
(After T. A. Rickard.)

rich patches, was worked on the main indicator and on parallel ones; but for the last 240 feet the former has been barren and

the latter have been much less productive than before. Samples were therefore taken on the 630-foot level, where the indicator had carried pay-quartz, and from the 770-foot level, where it had proved almost barren.

The ground near the indicators has been much faulted; and this faulting was found to have an interesting connection with the genesis of the auriferous deposits. The questions raised by Mr. Lidgey comprised, besides the general inquiries,—Was the gold on the indicators leached from the surrounding country-rock? Was it brought from below by ascending, or from above by descending, solutions?—the more specific ones, Why is the gold on these indicators concentrated in patches, and what effect have what are known to the miners as "heads," "headings" and "breasts"? Mr. Lidgey does not explain these terms; but they are probably related to the phenomenon observed by me in the Prince Regent mine, and reported to me as occurring in other mines in the belt, namely, that the chief quartz-veins crossing the indicator end always against a vein of "pug," or soft clay, filling a fault, along which more or less vertical movement has taken place. These slides, cutting off the quartz-veins, have, in my opinion, an important bearing on the questions under consideration.

Samples were taken in the Prince Regent mine: (1) of solid slate and sandstone, both close to, and as far as possible from, the indicator-beds known as "The Indicator" and "The Mundic Slate"; (2) of "pug," clay or broken quartz from the backs or breasts against which the veins crossing the indicator end; (3) of the quartz of the indicator-lodes, both containing and not containing sulphides; (4) of the indicator itself. The results of the analyses of these samples is shown in Tables IV. A and IV. B.

As these tables show, the solid country-rock, containing no sulphides, taken at whatever distance from the lode, is not auriferous; and this was equally true whether the veins on the indicator were productive or barren on the tract from which the sample of country-rock was taken. Moreover, sulphides are found in the country-rock only in close proximity to fault-planes or "backs." Finally, the "pug" filling these "backs" or slight fault-fissures is often highly auriferous, especially when it contains sulphides or an unusual percentage

of organic matter; and it was found to be as highly auriferous at levels where the indicator was barren, as where the indicator carried pay-quartz.

These results point to the following conclusions:

1. The gold on the indicator has not been leached out of the surrounding rock, but has most probably entered the lodes crossing the indicator, from below.

2. The channels that carried the upward currents from which the gold was deposited were probably the fault-planes or "backs," which cut off the indicator-veins.

3. The ascending solutions probably brought with them the gold from below, and gold was deposited along the indicators where the composition of the latter favored precipitation.

I believe therefore that the difference between barren ground and highly auriferous ground on an "indicator-line" is connected with some difference in the character of the indicator itself, at the two levels compared; and, in my opinion, the key to the distribution of gold in this most interesting ore-deposit should be sought in a careful examination of the various indicators which are found to be associated with auriferous and non-auriferous veins of quartz.

The Northern Star Co.'s Mine.—This mine is in Ballarat West, and an unusually good typical cross-section of the strata of that belt is afforded by a prospecting cross-cut, driven west at the depth of 1250 feet, and 1264 feet long in 1894 (since extended to about 1700 feet). At 100 feet from the shaft the Guiding Star lode was cut, and a level was driven along it for about 350 feet. The cross-cut traversed alternate beds of sandstone and slate, including several small reefs of barren quartz (see *k*, *q*, *y*, Table V.). At 1060 feet from the shaft, an eruptive dike 36 feet wide, and interbedded with the slates and sandstones, was driven through. It was felsite porphyry, much decomposed, and impregnated, especially near its contact with the country on each side, with much pyrite—the cubes of which were occasionally 18 mm. in diameter. This occurrence is specially interesting, since eruptive dikes are much rarer in the Lower Silurian than in the Upper Silurian auriferous area of Victoria. Samples were taken from this dike, and also at various distances along the cross-cut. Table V. contains the results of analyses, which are graphically shown in Diagram 3.

The remarks previously made on Tables I., II. and III. (Bendigo field) apply equally well here; but the analyses of the eruptive dike are specially interesting. The auriferous character of the pyrite in it (see "Dike 1 and 2," Table V.) indicates that an auriferous lode may be associated with the dike not far from its intersection by the cross-cut.

Steiglitz, Victoria.

This auriferous area, about 35 miles S.E. of Ballarat yielded, in the early days of quartz-reef mining, fabulously rich returns from its surface-quartz, and has again come into prominence within the last few years. It occupies an inlier of Lower Silurian slates and sandstones, surrounded on all sides by newer volcanic rocks. The country-rock shows signs of very extensive folding and faulting.

The United Albion Mine.—This is one of the deepest in the district, and has yielded the richest returns. Samples were taken from a cross-cut in the 850-foot level, which extends from the shaft through slates, with occasional interbedded sandstones, for 308 feet. The strata dip steeply towards the shaft until at 220 feet an anticlinal is reached (see *k*, Table VI.). At 68 feet from the shaft, the footwall of the Albion reef is cut (see *d*, *e*, Table VI.). The results of analyses are given in Table VI. and plotted in Diagram 4.

The Tam O'Shanter Mine.—This mine, though not very deep, is interesting by reason of an extensive fault in the strata. Samples were taken from two cross-cuts, one 260 feet below the surface, and running west from the shaft 185 feet; the other an east cross-cut, about 100 feet long, at a depth of 430 feet. The upper cross-cut starts in slates, dipping steeply west, but passes at 31 feet from the shaft a synclinal axis (see *i*, Table VI. A), beyond which the slates dip steeply E., until, at 83 feet, a fault-fissure, carrying broken slate and quartz (see *e*, Table VI., A), is encountered, dipping W.; and beyond this the strata (now alternating slate and sandstone) dip steeply W. again. The lower or east cross-cut starts in sandstones, with occasional thin beds of slate, all dipping steeply west. At 61 feet, in sandstone, it cuts the gold-bearing vein, dipping steeply E. The dip of the strata does not change beyond this vein, and the cross-cut ends in slate. The results of analyses are

shown in Table VI. A. The remarks upon the previous tables, relative to the country-rock of Bendigo and Ballarat, apply to Tables VI. and VI. A also.

Walhalla, Victoria.

One of the long narrow belts which constitute the auriferous areas of Victoria extends from Jamison on the north to Walhalla on the south, and is notable not only for the richness of some of its reefs, but also as presenting Upper Silurian instead of the Lower Silurian strata, and, above all, because of the peculiar relation between its pay-reefs and dikes of igneous rock—no productive reefs having been found in this belt or in any Upper Silurian area that were not associated with dikes; whereas in the Lower Silurian (Bendigo, Ballarat and Steiglitz) dikes are less common, and when they occur are usually found to be of more recent date than the reefs. (See R. A. F. Murray's "Geology and Physical Geography of Victoria," Melbourne, 1887.) The gold-veins in the Upper Silurian area are sometimes found crossing, more or less horizontally, dikes of great width, and dying out in the slates and sandstones on each side of the dikes. Another type of occurrence is illustrated by the eruptive dike of Cohen's reef, Walhalla, where the reef accompanies the dike, lying on one or both sides of the dike-material, between it and the enclosing country, or occasionally bounded on both sides by the dike. This mine was sampled, as will be observed further on.

Some writers have derived the auriferous contents of these lodes from the dikes.* But heretofore, so far as I know, no observer has actually analyzed samples of the dike to ascertain whether it is auriferous. Two other hypotheses are conceivable. On the lateral-secretion theory, the gold and silver of the quartz (and of the dike, if it carried any) might be derived from the country-rock on both sides; and, on the ascension-theory, the gold and silver might be derived from a deeper source.

The analyses undertaken in order to throw some light on this subject were: (1) of dike-rock containing no sulphides, and not associated with an auriferous quartz-reef; (2) of dike-

* See a paper "On the Formation of Mineral Veins and the Deposits of Metallic Ores and Minerals in Them," by H. A. Thompson, *Trans. Roy. Soc. of Victoria*, viii., pp. 228–249.

rock associated with such reefs; (3) of country-rock from the deep levels at various distances from the lode.

The Long Tunnel Mine on Cohen's Reef.—This mine is entered by a tunnel (see Fig. 2), at the end of which a shaft descends vertically 900 feet. From the bottom of this shaft a level (called the Northwest level in Table VII. A) extends 1800 feet, and from the end of this a second or main working-shaft descends about 800 feet. The working-levels from this shaft are 1022 feet, 1122 feet, etc., below the tunnel-level, and about 700 feet more than that below the surface of the hill. The Northwest level cuts through a barren dike and some quartz-veins. The dike associated with the main reef is exposed by a cross-cut from this level, and in the deeper workings below this level. Tables VII. A and VII. B give the analyses of samples from the two kinds of dike-material.

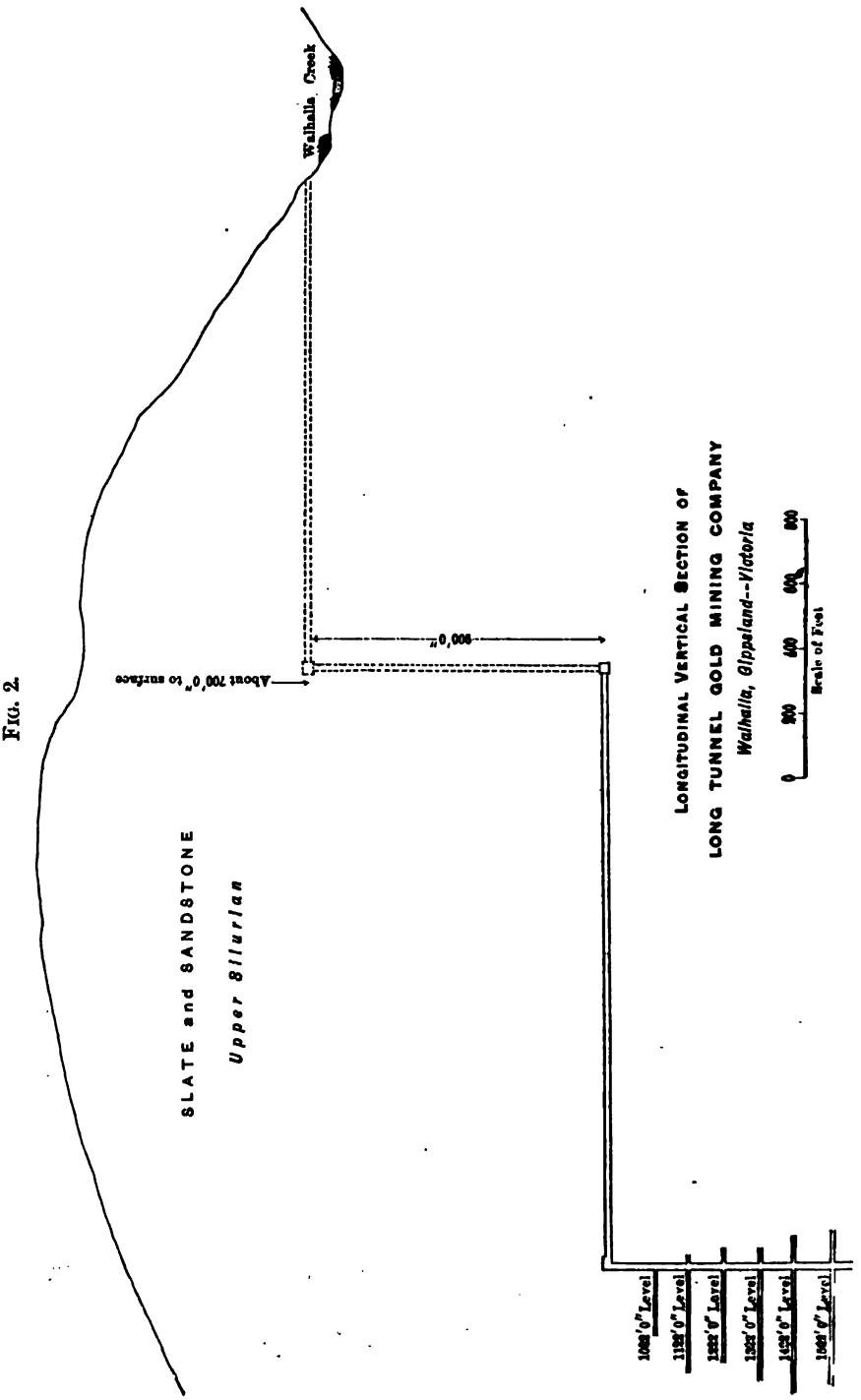
On the 900-foot level, a cross-cut 278 feet long has been driven W. to the dike. Samples of country-rock taken in this cross-cut are tabulated in Table VIII.; and Table IX. shows the results of samples from a cross-cut in the 1422-foot level, driven 165 feet east from the dike to the boundary of the claim.

Tables VII. A and VII. B show that, where no quartz is associated with the dike, the rock contains little or no sulphides, and is not auriferous; that where the dike is associated with quartz, but the quartz is not auriferous, the dike-rock is likewise barren; that the percentage of sulphides in the dike is greatly increased when it is associated with highly pyritiferous quartz, and that in this case the dike-sulphides are auriferous, though much less so than those of the associated reef.

As all the samples were taken from one dike of each class, it would not be safe to generalize too positively; but the results indicate that, in the Upper Silurian area, the dike-rocks themselves are not auriferous, but that their silver and gold are derived from the accompanying quartz-reefs, and that the agencies which filled with quartz the fissures in the dikes, impregnated the latter with auriferous sulphides—these being, however, less highly auriferous than the pyrite of the reefs.

Microscopic examination of samples of both barren and auriferous dike-rock from this mine showed that every sample was more or less decomposed and altered. The principal difference

FIG. 2.


Walhalla Creek
About 700'0" to surface
0,000"
SLATE and SANDSTONE
Upper Silurian
LONGITUDINAL VERTICAL SECTION OF
LONG TUNNEL GOLD MINING COMPANY
Walhalla, Gippsland--Victoria
Scale of Feet
0
200
400
600
800
1000'0" Level
1150'0" Level
1300'0" Level
1450'0" Level
1600'0" Level

clearly exhibited is the much greater abundance of pyrite and arsenopyrite in samples associated with an auriferous lode.

Concerning Table VIII., it will be noted that the samples are remarkable for the almost total absence of sulphides. Gold was found in only one sample from this cross-cut, and this sample was taken 97 feet from the dike. In this connection it is perhaps significant that when the dike was cut on this level, it was not associated with auriferous quartz.

The analyses given in Table IX. from the 1422-foot level correspond generally with those obtained from the deep levels of Bendigo; the proportion of sulphides and of gold increasing greatly as the lode is approached. The microscopical examination of thirty thin sections of country-rocks brought out very clearly the increasing development of quartz and pyrite as a lode was approached, but revealed, in most cases, nothing else of definite importance.

The Gympie Field, Queensland.

Apart from Mt. Morgan (which, being a single mine only, can scarcely be classed as a gold-field) the two most important gold-fields of Queensland are Gympie and Charters Towers. The Gympie rocks have been determined by Mr. J. Etheridge and others as Permo-Carboniferous. This field, which is only about two miles long and three-quarters of a mile wide, is remarkable for the evidence it affords of volcanic disturbance, contemporaneous with the formation of the sedimentary rocks. This disturbance has also probably produced the extensive faults intersecting the country-rock in various directions, and bounding the auriferous area on the north, south and west.

Mr. W. H. Rands, in his "Report on the Gympie Gold-Field" (Brisbane, 1889), says that the district presents a series of alternations of sedimentary strata with some intrusive igneous, and also some volcanic rocks. Taking the average dip at 22°, the aggregate thickness of the strata must be somewhat over 2000 feet, consisting chiefly of graywackes, altered sandstones, gray and black carbonaceous shales, grits, conglomerates, breccias and limestone. Interbedded with these are amygdaloid volcanic rocks and volcanic ash, and sheets of a much altered intrusive "greenstone." Most of the sedimentary rocks are more or less pyritous, some of the shales, espe-

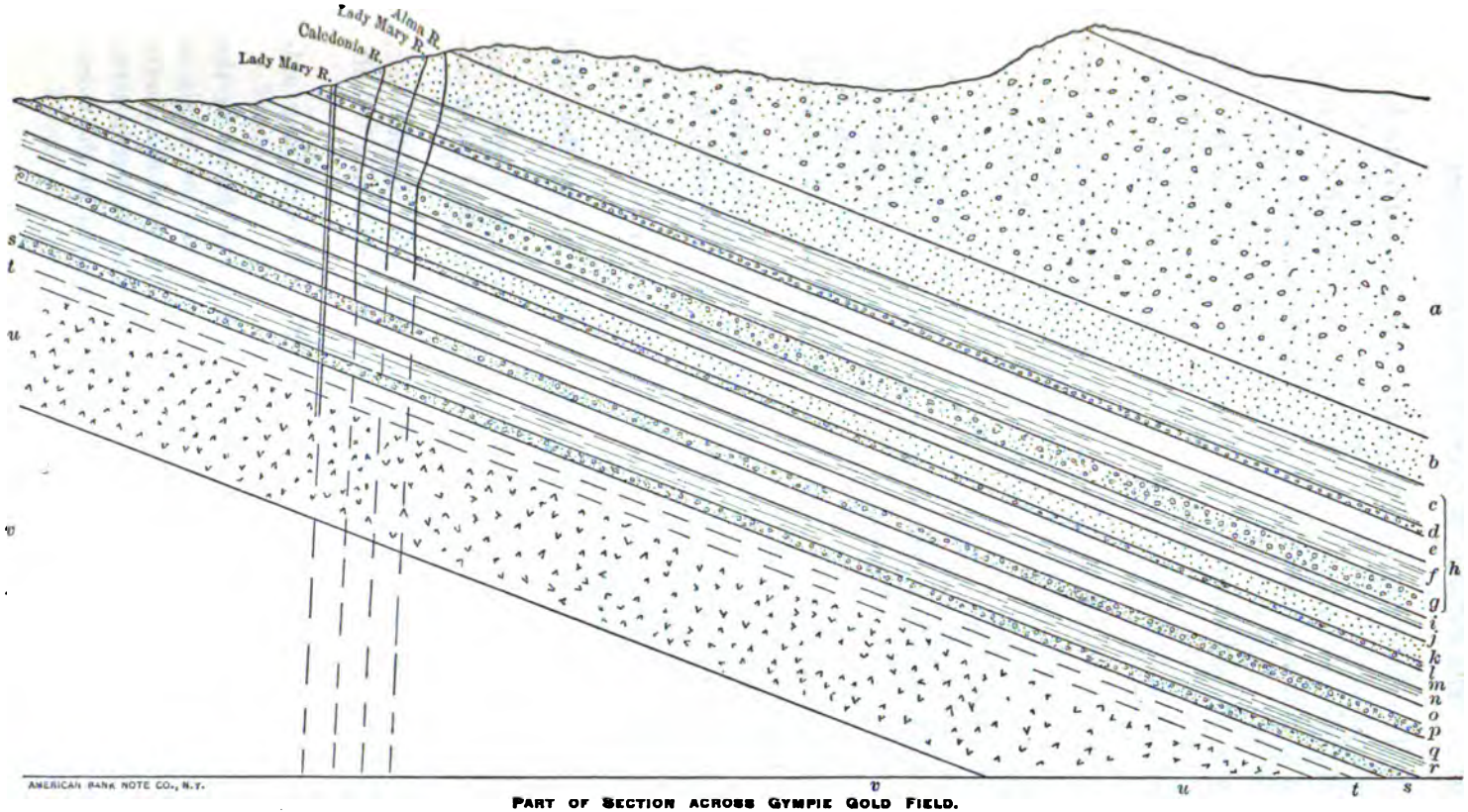
cially, being full of small cubic crystals of pyrite. Most of them, especially the sandstones, graywackes and conglomerates, and some of the shales, are also calcareous. The faces of joints are almost invariably coated with calcite. The strata dip with great regularity on the average from 20° to 22°; the direction of dip being, as a rule, in the northern part of the field, a little N. of E.; in the southern part, E.S.E. There are altogether four beds of dark carbonaceous shale in the formation, locally known as the First, Second, Third and Fourth beds of slate; and it has been observed that the reefs are notably profitable only where they traverse these shales. I found in the latter, as a mean of four determinations, 1.07 per cent. of carbon; but I did not find the percentage of carbon increasing as a lode was approached—as was the case in the New Chum Railway mine of Bendigo, described above. They generally carry in very small cubes a considerable quantity of pyrite—the specimens of other sedimentary rocks of the district examined by me being much less pyritous. The intrusive "greenstone" interstratified with the shales, appears to have affected favorably the gold-bearing character of the rocks near it. Mr. Rands, while pointing out this favorable effect on the shales, says, however, that the reefs actually in contact with the "greenstone" have become impoverished.

The reefs, as shown in a section accompanying the report of Messrs. R. L. Jack and R. Etheridge on the "Geology and Palæontology of Queensland and New Guinea" (Brisbane, 1892), are fissure-lodes, cutting the series of strata nearly vertically. The section referred to is reproduced in Fig. 3, in which the different strata are lettered as follows:

a, Coarse, pebbly conglomerate, with beds of sandstone; *b*, Laminated sandstone; *c*, Shales (fossiliferous); *d*, Coarse, pebbly conglomerate; *e*, Shales and thin-bedded sandstone (fossiliferous); *f*, Shales (fossiliferous and hard gray sandstone); *g*, Coarse, pebbly conglomerate; *h*, Phœnix or Upper shales, comprising *c*, *d*, *e*, *f*, and *g*; *i*, Shales; *j*, Angular grit (probably volcanic ash); *k*, Greenish, highly fossiliferous sandstone; *l*, Conglomerate; *m*, Green crystalline rocks; *n*, Shales ("First bed of slate"); *o*, Fine grained, hard gray sandstone; *p*, Altered coarse, pebbly conglomerate; *q*, Altered graywacke (semi crystalline); *r*, Shales ("Second bed of slates"); *s*, Coarse, pebbly conglomerate; *t*, Porphyry (probably diabase-porphyry); *u*, Hard, crystalline greenstone; *v*, Green and purple chloritic rock, in parts amygdaloidal; cavities filled with carbonate of lime (volcanic, probably an amygdaloidal diabase).

The questions suggested by the foregoing conditions are:

FIG. 3.



PART OF SECTION ACROSS GYMPIE GOLD FIELD.

Showing kindly shales and associated eruptive rocks

Copy of section by R.L.Jack.

Scale: Horizontal and Vertical 200 Feet=1 Inch.

1. The shales being evidently the favorable country-rock, was the gold leached out from them, carried into the lode-fissures and there precipitated? In this connection, it will be interesting to note whether the pyrite in the shales carries gold both near and away from the lode, and whether the non-pyritous shales are gold-bearing. Moreover, in view of the favorable influence of the "greenstone," is the source of the gold to be found in the silicates, pyrite or other minerals of the "greenstone"?

2. If the country-rock is found to be auriferous, is the gold in it, or in the pyrite or other mineral which it contains, derived from the lode, which has in turn received it from a source deeper than any of the rocks exposed?

Samples collected from the North Phœnix mine, by Mr. Alfred Lord, of Gympie, were analyzed with the results shown in Table X. and Diagram 5. While this number was not as great as I could have wished, the samples taken were fairly typical of the various classes of rock, and the table indicates the following noteworthy features:

1. The favorable country-rock (dark carbonaceous shale) is auriferous only where it carries pyrite, and especially in the vicinity of the lode.

2. The percentage of pyrite is much greater near the lode than at a long distance from it.

3. The pyrite near the lode is much richer in gold than that at a distance.

4. Even "unkindly" country-rocks, such as purple conglomerate and greenstone, when near the lode, and when carrying pyrite, are auriferous.

5. Both the kindly shale and the unkindly country-rock, when taken from the vicinity of the great "Phœnix" fault, carry pyrite, which is auriferous.

6. No gold was found in any sample not containing sulphides.

These results seem to indicate that the gold found in the country-rock is derived from the lode, and that the origin of the gold in the lode is to be looked for neither in the kindly country-rock nor in the greenstone underlying it, but in some source lying deeper than any rocks now exposed in the workings.

Charters Towers, Queensland.

This district, which stands first among the Australian producing areas, affords an interesting example of rich auriferous quartz-reefs in massive crystalline country-rock. Mr. Rands, in the report already cited, says that the richest part of the gold-field lies on the western edge of a large granitic area; the granite varying from a type in which orthoclase, mica and quartz are the essential minerals, with hornblende as an occasional accessory, to a type in which, besides orthoclase and quartz, hornblende is essential and mica occasional. The samples furnished to the writer by Dr. Redmond, of Charters Towers, were, however, all quartz-diorites, with mica in very small proportions, and have been determined by Prof. Ulrich, of the Otago School of Mines, as varieties of tonalite. In the Brilliant and St. George Co.'s mine, from which the samples were taken, this quartz-diorite formed both foot- and hanging-wall of a quartz-reef which yielded on assay an average of 2 oz. 6 dwt. 9 gr. of gold, and 13 dwt. 4 gr. of silver, per ton. The samples were taken from a cross-cut 1079 feet from the surface, and show a remarkable similarity, irrespective of distance from the lode, with one exception (*a*, Table XI.), which was taken but 2 feet from the lode, and carried a vein of auriferous pyrite.

An account of the separation and analysis of various crystalline silicates from these specimens is given further on, in the section dealing with the analysis of the constituents of crystalline rocks. The results of that inquiry confirm the conclusion drawn from Table XI., that the country-rock contains neither gold nor silver. The exception (*a*) above noted, was almost certainly due to an impregnation from the neighboring lode.

Macetown, Otago, South Island of New Zealand.

The country-rocks of the Otago gold-veins are metamorphic, gradually changing from true phyllite in the east to much contorted and foliated mica-schists towards central Otago. In the *N. Z. Geological Report* of 1878–79 (p. 14) Sir James Hector arranges these mica-schists in three divisions, in the middle one of which lie the most productive gold-fields of Otago.

These schists contain numerous quartz-folia, mostly not more

than 1 or 2 inches thick, but occasionally reaching 8 inches or a foot. In the latter case, they form lenses of quartz, not usually of great extent in strike or dip.

The quartz-lodes, which usually dip steeply, cut across the bedding of the schists.

The district of Macetown is in the heart of a mountain region, 20 miles N.E. of Queenstown, on Lake Wakatipu. Here well-defined lodes are worked by means of tunnels which reach a depth below the surface of at least 1500 feet. Samples to illustrate the deep underground circulation were taken from two mines named below, with the assistance of Mr. W. J. Stanford, the general manager of both.

The Tipperary Mine.—A specially good cross-section of the country at right angles to the strike of the reef is afforded by the prospecting deep adit of this company, which was started 1840 feet from the footwall of the Tipperary lode, and intended to strike the lode in a part which had proved rich in the workings above. At the time I took my last samples, the adit had been driven 1530 feet. At 164 feet from the mouth, the Dublin United reef, a gold-bearing lode 1 foot wide; and 876 feet further (or 940 feet from the mouth) another, the Killarney reef, 1 foot 6 inches wide, was crossed. The Tipperary reef was known to be about 300 feet further on. Over 70 samples were taken from this adit, the farther end of which was approximately 1000 feet below the surface. The samples taken between the mouth and the Dublin United reef represent the vadose circulation, and will be considered later under that head. The remainder form a very good typical series from the deep circulation.

Special interest attaches to the analysis of the quartz-folia already mentioned as abounding in the mica-schists of Otago. In this adit a number of such folia of fair width were crossed at various distances from the two lodes cut by the adit, and samples of these, as well as of mica-schist not containing quartz-folia of unusual size, were taken.

If gold be disseminated through the mica-schist country-rock, and the gold of the lodes be due to a leaching of this rock, there seems to be no reason why these quartz-folia—each of which constitutes, as it were, a miniature lode—should not

have been similarly enriched from the rock on both sides, especially as the solvent action of water percolating through sedimentary rocks during their metamorphosis would almost certainly be greater than that of the water which carried silica in solution and formed the quartz-reefs. The detection of gold in these quartz-folia, at a long distance from any auriferous lode, would therefore give strong support in this case to the lateral-secretion theory, and *vice versa*.

Table XII. contains the results of analyses of such quartz-folia from the Tipperary deep adit, together with a few similar samples from the Premier mine. It shows that the quartz-folia in this mica-schist are not auriferous when sulphides are not present, and that sulphides are present in appreciable quantity only when the quartz-folia are near an auriferous lode.

Table XIII. contains the results of analysis of the country-rock in the Tipperary deep adit, which are plotted in Diagram 6.

The Premier Mine.—This mine is situated a few miles from the Tipperary. In both this and the Tipperary mine a great deal of work has been done along the lode and at short distances from it. The lode-fissure is clearly traceable for the whole distance (in one case over 2000 feet) to which it has been drifted on; but the greater part of it is filled with "mullock," or broken mica-schist, with small quartz-veins, the whole being often fantastically contorted, and containing, as a rule, much carbonaceous matter. The quartz of the lode runs in shoots of varying width, only one of which has been worked in the Tipperary mine, while two have been worked in the Premier. For the purpose of studying the rock near the reef, samples were taken from short prospecting cross-cuts run from the lode in various directions, as well as from parts of the mine still nearer the lode-fissure. In the latter case care was taken to select samples from parts of the mine opposite gold-bearing quartz, and also from parts opposite places where the lode carried mullock, and not auriferous quartz. The results are given in Table XIV. and plotted in Diagram 7.

Tables XIII. and XIV. illustrate again the striking difference, already noted, in both the yield of gold and the percentage of sulphides, between samples taken near lodes and those taken at a distance.

Reefton, Nelson, South Island of New Zealand.

This district, till lately the most productive of gold from quartz-lodes in New Zealand, carries its gold-veins in Carboniferous rocks; the Devonian strata, though well represented in it, not having been found to contain auriferous reefs.

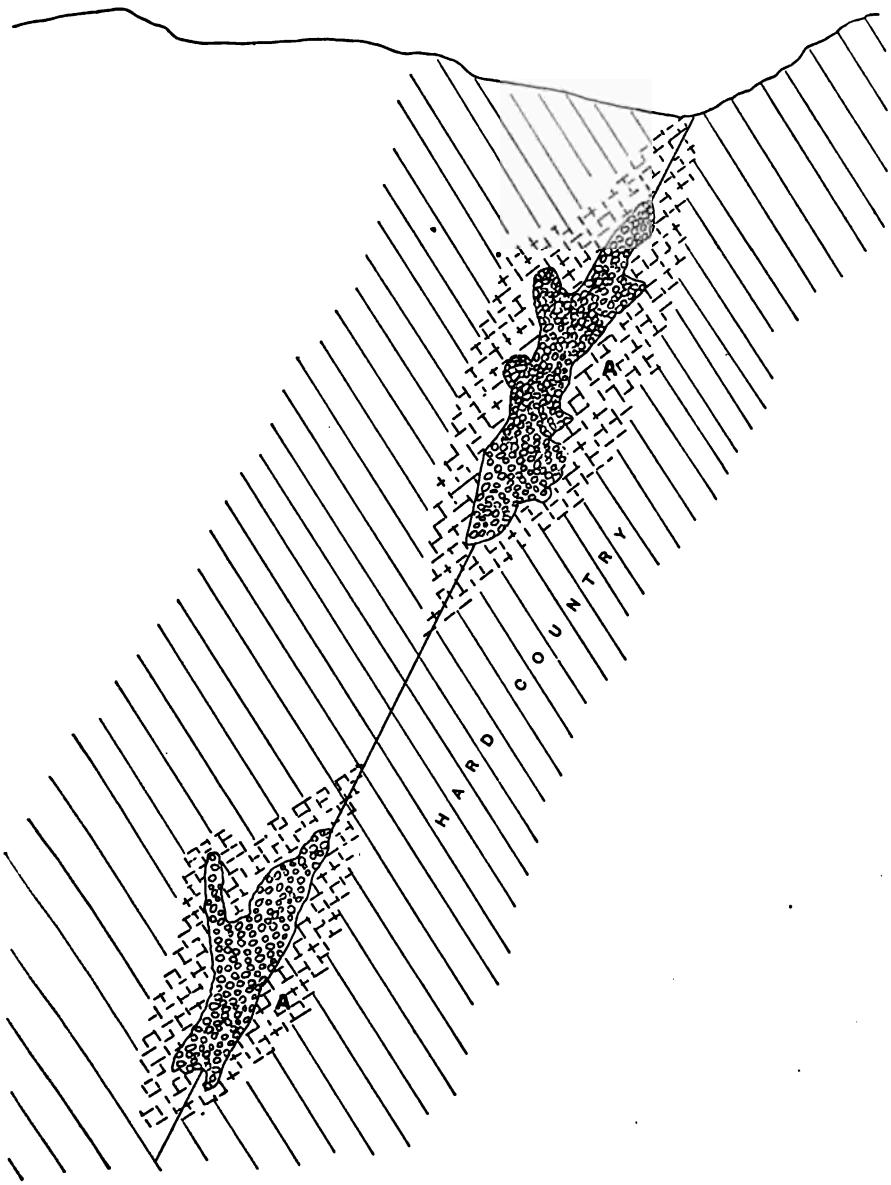
The Reefton quartz-bodies often consist of bunches or irregular masses, branching in all directions into the country-rock. These are aggregated along certain lines, the space between two bunches on such a line being occupied by a connecting-fissure, generally very narrow and often filled with quartz or "pug." (See Fig. 4.)

The bunches of quartz are not continuous to any great extent. The horizontal section of one of them has been compared in form to an octopus. It is noticeable that the lodes pinch to a mere trace or connecting-fissure in the harder parts of the country-rock, while the quartz-bodies are invariably surrounded by rock showing evidences of severe faulting and crushing. Nearly every sample of this softer country-rock analyzed was traversed by slickensides and heads in all directions to such an extent that it was difficult to procure a solid hand-specimen, as the pieces would not stand dressing with the hammer. Samples illustrating the hard rock enclosing the narrow lode-fissures, and the softer, kindly, much-faulted rock bounding the wider auriferous bodies were taken by Mr. T. Esdaile, of the Otago School of Mines, from the deep levels of three of the best-developed mines of the district—the Wealth of Nations, the Progress and the Hercules. The results of analyses are given in Table XV., which shows: (1) that the hard, solid slate contains, as a rule, very little pyrite, and is not auriferous; (2) that the broken country around the bunches of auriferous quartz contains a much larger percentage of pyrite, and that this pyrite, when near the lode, is auriferous; and (3) that although several samples of slate taken 140 to 200 feet from the lode carried large quantities of pyrite, this pyrite contained no gold (see *e*, *g*, *m*, Table XV.).

Thames, North Island of New Zealand.

This gold-field, in the Coromandel Peninsula of the North Island, presents a new type of country-rock, as compared with those considered above, all of which (except that of Charters

FIG. 4.



VERTICAL SECTION TO ILLUSTRATE BUNCHES OF QUARTZ
OF
REEFTON GOLD FIELD.
A--Soft broken country
[NOTE.—This section is merely diagrammatic.]

Towers) were sedimentary strata. The country-rock of the Thames district (probably, for its area, the richest ever yet

worked) consists of alternating hard and soft bands of hornblende- and augite-andesites, of Lower Tertiary age. Auriferous lodes occur in both hard and soft (decomposed) andesites, but they are not profitably rich, except where they traverse the latter. The undecomposed rock is extremely hard, and usually greenish or dark blue. It is locally called "bluestone." The decomposed rock varies from almost pure white to gray and brown, and is often highly impregnated with pyrite. Its local name, "kindly sandstone," roughly indicates its appearance. The name propylite, proposed by Richthofen for an altered andesite like this, and applied to the "kindly country" of the Thames by Mr. James Park,* lately Director of the Thames School of Mines, is similarly used in the present paper. This propylite contains a large number of auriferous lodes, practically parallel, and generally striking N.E. The gold they contain is usually alloyed with 30 to 40 per cent. of silver. In some parts of the field, the silver constitutes 50 or 60 per cent. of the bullion obtained.

Prof. Hutton has shown† that the propylite consists entirely of the products of decomposition of the unaltered andesite with which it is associated. (Becker, in his work on the Comstock Lode, has shown a somewhat similar decomposition of the andesite and diabase which bound it). Prof. Hutton gives it as his opinion that the origin of the gold in the auriferous lodes will probably be found in the pyroxenes of the andesites, and strongly advises chemical examination of the andesites to clear up this question. Mr. Park, in his report already cited, agrees with Prof. Hutton, and promises further investigation. Both these authors ascribe the alteration of the hard andesite into propylite to heated waters, carrying hydrogen sulphide and other acid vapors. These waters, having a high temperature, must have been, of course, ascending waters. This narrows the inquiry to two questions:

1. Do the gold and silver of these lodes form a part of the crystalline constituents (pyroxene or other silicates) of the original andesite, and did the heated waters, decomposing these

* See Mr. Park's admirable report on the Thames gold-fields, printed as an appendix to the New Zealand *Mining Report* of 1893.

† "On the Rocks of the Hauraki Gold-field," by F. W. Hutton, F.G.S., *Austral. Ass. Adv. Sci.*, vol i., 1887, pp. 245–274.

crystalline constituents, dissolve the gold and silver and carry them into the lode-fissures where they are now found? or,

2. Did the heated waters obtain their gold and silver from rocks at greater depths, whence they were carried into their present position?

If the first question is to be answered in the affirmative, then analysis of the unaltered rock, and especially separate analysis of its isolated crystalline constituents, should detect gold and silver. Otherwise, it seems reasonable to conclude that some such hypothesis as is stated in No. 2 is the true one. So far as I am aware, no analysis of the crystalline constituents of the Thames andesites has been made hitherto, though Prof. Hutton mentions that in one instance gold has been found in pyrite not actually taken from the lode itself.

The Moanataiari Tunnel.—A very good section of both the unaltered andesite and the propylite, as exposed in a deep level, away from the oxidizing influence of the ground-water, is afforded by the Moanataiari tunnel, which has been driven 3125 feet into the Kuranui hill, and is 1530 feet beneath the surface at its inner end. This adit crosses several belts, both of hard and of kindly rock. The gold-bearing veins intersected by it are in many cases associated with the latter. At 1500 feet from the mouth, the great Moanataiari fault was struck, bringing hard unaltered country opposite the kindly propylite.

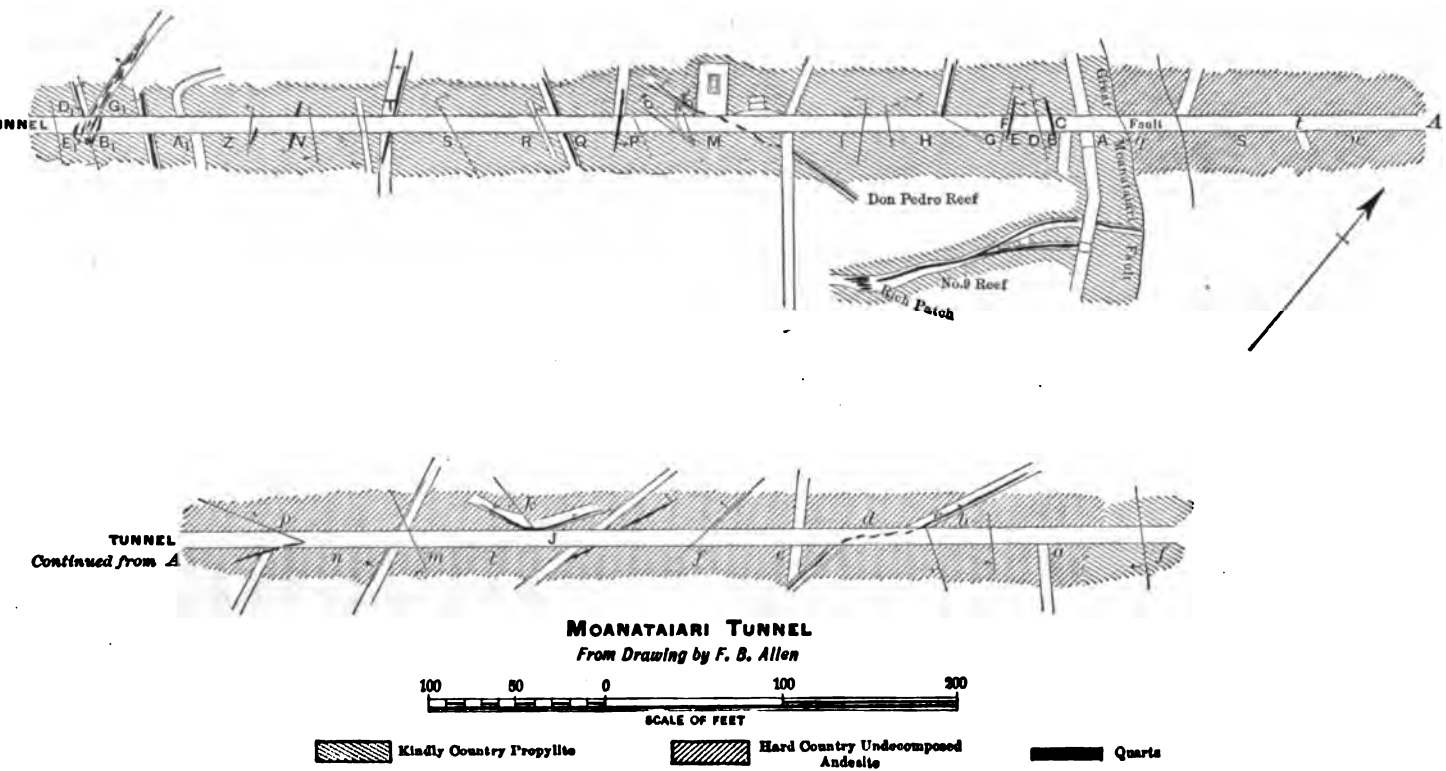
Tables XVI. and XVII. contain the results of analyses, taken as indicated in the plan of the tunnel, Fig. 5. In every case in which the sample was concentrated, the silver as well as the gold in the concentrate was determined; and the usual examination for insoluble sulphides was made in this field as in all others.

Tables XVI. and XVII. exhibit a striking difference between the hard andesite and the kindly propylite, both as to their gold and silver, and as to their sulphide contents. Taken together with Table XVIII., they point to the conclusion that the gold and silver of the propylite are not derived from the pyroxene or from any other constituents of the unaltered rocks, but must have their origin in some rock not reached by any of the present mine-workings.

Prof. Hutton* thinks there "can be no reasonable doubt that

* *Op. cit.*, p. 265.

FIG. 5.


TUNNEL
Don Pedro Reef
No.9 Reef
Rich Patch
Fault
Moanataiari Fault
TUNNEL
Continued from A
MOANATAIARI TUNNEL
From Drawing by F. B. Allen
100 50 0 100 200
SCALE OF FEET
Kindly Country Propylite
Hard Country Undecomposed Andesite
Quartz

the gold came out of the volcanic rocks, and was not brought into them from below," and gives five reasons for this opinion. But he admits that no chemical examination of the country-rock has been made, and also (on p. 269) that Sir James Hector, who has had large experience of this field, believes that the gold and silver came from a source lower than the andesites which bound the reefs. The evidence supplied by Tables XVI., XVII. and XVIII. should throw additional light on the subject.

It will be noticed in Table XVI. that there is a great difference in fineness between the gold in the favorable country-rock and that found in the quartz lodes running through that rock. Seven samples taken from various reefs crossed in the tunnel gave an average fineness of 63.18 gold and 36.82 silver, while the average fineness of the bullion obtained from the pyrite of the country-rock is 22.93 gold and 77.07 silver. Considering the fairly large number of samples analyzed, this difference can scarcely be regarded as accidental. It will be interesting to note whether future analyses of the pyrite from the propylite confirm it.

Chapter V.—The Examination of Various Constituents of Crystalline and Eruptive Rocks for Gold and Silver.

In the South Island of New Zealand an unusually favorable opportunity is offered for the analysis of the older crystalline rocks, underlying the sedimentary rocks which form the "country" of the gold deposits. The Manipori formation of this island, comprising the greater part of the mountainous district west of lakes Manipori and Te Anau, in Otago, consists of an enormous thickness (estimated by Prof. Hutton* at 160,000 feet) of crystalline schists, gneiss, syenite and syenitic gneiss, with associated masses of granite; the whole forming the most picturesque part of New Zealand. The famous West Coast sounds occur in this formation. No paying auriferous lodes have been discovered in it; and for this reason an examination of the rocks is specially interesting here. Stelzner, Posepny and others, who have criticized the conclusions of Sandberger, have laid great stress on the fact that all the silicates analyzed by him were taken from the vicinity of ore-

* *Geology of Otago*, by Hutton and Ulrich, p. 28.

bodies containing those heavy metals which he notes as occurring in the silicates of the country-rock; their contention being that these metals, supposed to occur as silicates in gneiss and other crystalline rocks, were really contained as sulphides, and were therefore impregnations from the neighboring ore-bodies. The same objection, whatever be its weight, might be urged with equal justice against Mr. Becker's derivation of the mercury and gold of the lodes of the Pacific Coast from the granite underlying their country-rock. In the case of the Manipori formation, just mentioned, such criticism could hardly be made, as no auriferous lodes have been found within many miles of the district from which samples were taken for the present investigation.

In the South Island also, samples were taken from the granite abutting on the Carboniferous strata near Reefton, which latter are the chief carriers of auriferous lodes in the western part of the island.

Other samples of granite were collected from various parts of Westland and Nelson, as far north as the granite quarries of Cape Foulwind.

Selwyn has pointed out* that the auriferous Silurian rocks of Victoria probably rest everywhere on granite; and this conclusion has been supported by the later observations of Murray† and others. This underlying granite has not been reached in any of the mines working in the stratified rocks about it; but it is exposed at the surface in many parts of the colony, both near to gold-bearing areas and distant from them. Numerous samples of it were taken for this investigation.‡

The study of the country-rock from Gympie and Charters Towers, Queensland, gave a good opportunity for the analysis of silicates from crystalline igneous rocks. In the case of Charters Towers, the whole country-rock (locally termed granite) is quartz-diorite, with a little accessory mica; while at

* *Geology and Physical Geography of Victoria*, by A. R. C. Selwyn and G. F. H. Ulrich.

† *Geology and Physical Geography of Victoria*, by A. R. F. Murray, Melbourne, 1887.

‡ In New South Wales, granite forms the country-rock of a number of auriferous lodes; and it is much to be desired that some investigator on the spot would take up this inquiry. The present paper contains no analyses of New South Wales granite.

Gympie a thick sheet of diorite aphanite (locally termed greenstone), containing a large percentage of carbonate of lime, is interbedded with the shales which bound the auriferous reefs, these kindly shales occurring both above and below it. Messrs. Alfred Lord and R. Steele, of Gympie, have furnished samples of the "greenstone."

Sandberger's results having been criticized on the ground that all the rocks he analyzed contained sulphides, and Posepny* having urged the same objection to Becker's conclusions in the case of the Comstock lode, this point was carefully guarded. In every instance (even though an examination of a hand-specimen showed no sulphides), an analysis of the rock was first made, to determine whether sulphides were present; and whenever such was the case, the sulphides were isolated by panning, and separately assayed for gold and silver; but *when sulphides were thus found, no separation of the other crystalline constituents of the rock was made.*

It was deemed necessary to isolate and examine the following minerals: (1) *Mica* and (2) *Hornblende*, because to these Sandberger traces the silver and gold of many European lodes; (3) the *Pyroxenes* of the later eruptives, because Becker traces the gold and silver of the Comstock to the augite of the diabase, and both Hutton and Park (already cited) are inclined to refer the gold and silver of the Thames district, N. Z., to the pyroxenes of the andesite;† and (4) *Magnetite*, because of Prof. Hutton's remark:‡

"I would suggest that, as part, at least, of the pyrites has been formed from magnetite, the gold may have been originally in the magnetite, and have been released during the formation of the pyrites. I do not think that this has been the case, *but it is a point worthy of investigation by the chemist.* The pyrites is, no doubt, a secondary mineral, formed in the rock after consolidation; and if it should turn out to be generally auriferous, we must suppose either that the gold came from below with the sulphur, or that its source is the titaniferous magnetite which is one of the original constituents of the rocks."

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* "The Genesis of Ore-Deposits," *Trans.*, xxiii, 282.

† Prof. Hutton says on this point (*op. c t.*, p. 272): "If, therefore, we assume that the pyroxenes of our volcanic rocks contain gold and silver, that the conditions necessary for dissolving them rarely obtain, but that one of the exceptions has been in the Hauraki [Thames] gold-fields, we have a hypothesis which will, I think, explain most of the facts."

‡ *Op. cit.*, p. 271.

Isolation of Minerals.

The crystalline rocks examined comprised: (1) those in which certain minerals were distributed in large crystals or arranged in folia, so that clean samples could be picked out by hand; and (2) those from which the various minerals had to be separated by means of a liquid of high specific gravity.

The first class included many samples from the gneiss, syenite and granite of the Manipori formation, N. Z., from which the separation of mica and hornblende was specially easy. In various parts of this formation mica-veins are abundant. It was sometimes possible to pick in a few minutes 2 or 3 pounds of clean mica in plates up to 2 or 3 inches square. The syenite and syenitic gneiss often contain bands of almost pure hornblende, quite an inch thick, permitting a similarly easy isolation. Finally, magnetite can, of course, be easily separated from any rock in which it occurs.

The treatment of the second class, comprising many of the rocks examined, may require some explanation.

The separation of minerals from rocks by means of heavy solutions is extremely tedious, and it is therefore desirable to reduce the quantity of the rock to be operated upon if this can be done without material loss of the mineral to be isolated. As all crystalline rocks contain a large percentage of material below 3 in sp. gr., which can be effectively removed by panning, each sample was concentrated first by panning to get rid of quartz, feldspar and other specifically light materials. This procedure would, of course, be inadmissible if the object had been to determine accurately the gold and silver *per ton of the rock examined;* but the purpose in each case being simply to get as large a quantity as possible of a given mineral, and to get that sample approximately pure, this preliminary crude concentration seemed to be unobjectionable.

As an illustration, the following record of the process, as applied in one instance (Sample 7, Table XVIII.), is here given. With suitable modifications, this method was followed in all similar cases:

Sample of Syenite.

1. The sample was roughly broken and examined with a good lens, but no sulphides were discovered. A chemical analysis of 10 grammes of the sample also showed that sulphides were absent.

2. A portion of 500 grammes, pulverized so as to pass a No. 30 sieve, was reduced by careful panning to 217 grammes, of which 58 grammes was removed by a weak magnet. The portion thus removed proved, under the microscope, not to be pure magnetite, but to contain a good proportion of hornblende and feldspar (chiefly orthoclase) grains, probably drawn to the magnet by small adhering particles of magnetite.

3. The 159 grammes not attracted by the magnet consisted chiefly of hornblende and feldspar, with a little mica (biotite) and quartz. This powder was introduced into an apparatus modeled on Thoulet's, but made of large size, to save time. The heavy liquid used was Sonstadt's: mercuric iodide, dissolved in excess of potassic iodide, and having 3.175 sp. gr.

On exhausting the air from the apparatus, 77.5 grammes of practically pure hornblende fell to the bottom of the liquid. A dilution of this liquid to 2.95 sp. gr. was followed by the precipitation of 35.5 grammes more—also nearly pure hornblende. The total of 113 grammes, reckoned as hornblende, was finely powdered and assayed with pure litharge. No trace of gold or silver was obtained.

4. The 58 grammes removed by the magnet was similarly powdered and assayed, with the same result.

Sonstadt's solution is cheap and easily prepared, but has the great disadvantage of being extremely corrosive. Even with the greatest care, it was found almost impossible for the operator to avoid burning his fingers. Hence, in a number of instances Klein's solution of boro-tungstate of cadmium was employed. This has another advantage over Sonstadt's solution, namely, that when a partial separation of hornblende from augite is desired, the greater part of the hornblende can be kept from sinking in the liquid by using the latter in the concentrated state.*

The results of the analysis of minerals separated from 47 samples of rock are given in Table XVIII. On this table the following additional notes are presented:

1. Sample 17 represents a rock of peculiar occurrence, occupying a narrow strip in an area of Tertiary trachyte at Portobello, on the east coast of the Otago peninsula. Samples taken short distances apart vary much in character; but, on the whole, the rock may be classed as a diorite. Gold was said to have

* It was found impossible to separate augite from hornblende completely by the use of heavy liquids; but since no gold or silver was found in either, this fact did not affect the practical results of this investigation. The same remark applies more or less to all the minerals thus isolated for analysis. The writer knows of no method which will perfectly isolate considerable quantities of one mineral from other minerals not widely removed in specific gravity. In Table XVIII., therefore, the terms "hornblende," "augite," etc., simply designate the greatest part of the mineral samples to which they are applied.

been found *in the rock*, up to half an ounce to the ton. If this were the case, such an occurrence of gold would have been, so far as the writer knows, unique.

Of the three samples chosen, one (No. 17) contained pyrite, and this pyrite was—but the rock itself was not—auriferous. The occurrence of auriferous pyrite in such an area (of recent volcanic rocks) is most unusual, and Prof. Ulrich, Director of the Otago School of Mines,* who has examined the locality and studied rock-sections from it, thinks that this dioritic rock probably underlies the basic volcanics which form the rest of the peninsula. In that case, it would be allied rather to the dikes of the Upper Silurian in Victoria than to the newer volcanic rocks of its immediate neighborhood.

A prepared thin section of this rock was microscopically examined by a German mineralogist, who says of it:

"This rock is, without doubt, one of the older volcanic rocks. It consists of feldspar, mica and hornblende, with a little quartz and magnetite. Mica is to a great extent absorbed, and magnetite has taken its place. The rock is difficult to classify, but would be best described as an elæolite-syenitic rock."

2. Samples 29 to 34 are from the Thames district. It was found difficult to get samples of the Thames andesites in which the analysis of 5 grammes would show no trace of sulphides—this being, as already explained, a requisite condition for the particular investigation in hand. A large number of samples had to be rejected on this account; but in Nos. 29 to 34 no trace of sulphides was found. Tables XVI. and XVII., on the other hand, give numerous samples of the Thames andesite country-rock, containing pyrite and other sulphides, carrying in many cases notable quantities of gold and silver.

3. Nine samples in Table XVIII.—namely, Nos. 6, 11, 13, 17, 26, 36, 38, 46 and 47—contained sulphides. The results of further examination were as follows:

No. 6. Sample of 500 grammes gave 53 grammes of sulphides, mostly pyrrhotite and chalcopyrite, which contained 0.0053 grain of silver.

No. 11. Sample of 500 grammes gave 18.56 grammes, chiefly pyrrhotite. No gold or silver.

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* I desire at this point to express my grateful thanks to our respected chief, Prof. Ulrich, for much kind encouragement and practical assistance.

No. 13. Sample of 1000 grammes gave 12.18 grammes of pyrrhotite and arsenopyrite. No gold or silver.

No. 17. Sample of 1000 grammes gave 4.934 grammes of pyrite, containing 0.037 grain of gold and 0.0041 grain of silver.

No. 26. Sample of 1000 grammes gave 28.92 of arsenopyrite and pyrrhotite, with a little galena, containing 0.0032 grain of gold and 0.0019 grain of silver.

No. 36. Sample of 1000 grammes gave 16.48 grammes chalcopyrite, with a small quantity of bournonite, carrying 0.407 grain of silver and no gold.

No. 38. Sample of 1000 grammes gave 16.03 grammes of pyrite. No gold or silver.

No. 46. Sample of 1000 grammes gave 8.42 grammes of pyrite. No gold or silver.

No. 47. Sample of 500 grammes gave 42.7 grammes of pyrite and galena, containing 0.0016 grain of gold and 0.0037 grain of silver.

Conclusions.—The results summarized in Table XVIII. were greatly surprising to the writer. In view of the usually tedious character of the operation of isolating the various constituents of a rock, he would not have examined so large a number of samples had he not expected, at each new analysis, that he might succeed in discovering gold in some mineral other than a sulphide.

It is, perhaps, comparatively easy to conceive why, in a stratified area, gold may occur only in connection with sulphides; but that in such a rock as gneiss, granite, syenite, or diorite, it should form no part of the crystalline constituents, but, on the contrary, should occur only in the sulphides found in these rocks, seems more remarkable and significant.

Chapter VI.—The Vadose Region.

Prof. Posepny* advances the sweeping proposition that the formation of ore-deposits could have taken place by descension and lateral secretion in the vadose region of circulation only, and must have been, in the deep region, the product of ascending currents. This distinction is, perhaps, too sharply drawn by him. It seems to the writer that the lateral-secretion theory

* *Op. cit.*, *Trans.*, xxiii., p. 262.

can scarcely be put out of court by assuming a lateral-secretion to be impossible below the ground-water level. Yet the marked difference everywhere observed in the contents of auriferous lodes above and below that level required that the rocks of the two zones should be distinguished, and separately analyzed in this investigation.

From an economic standpoint, this difference is expressed by the almost universal experience in the Australasian gold-fields, that the average yield of gold is much smaller below the water-level than near the surface. This statement, which will doubtless be controverted in some quarters, is based on the concurrent testimony of a large number of mine-managers and others, having long experience in the auriferous deposits of Australia and New Zealand. The almost unanimous evidence is in favor of the greater richness of vadose deposits. Several men of great experience have even given the opinion that, for ounces per ton above the ground-water level, only pennyweights per ton have been found below it.

This important economic question is naturally discussed in treatises on ore-deposits. Phillips,* for example, gives a number of reasons why the results in the vadose region may seem to be, while they are not really, higher than those of deep levels. Even after taking these considerations into account, however, the evidence of greater richness in the vadose region in Australia seems overwhelming.

In this connection, reference may be made to the exhaustive work of Mr. R. Brough Smyth,† and to a very interesting little work,‡ dealing with the yield of the auriferous deposits of Victoria from 1851 to 1862. The reader of the latter book may suspect the anonymous author of overstating the facts; but a comparison of the average yields noted in it with those of mines now working in the same reefs, with the aid of the latest gold-saving appliances, can hardly fail to carry conviction, even to those who, permitting "the wish" to be "father to the thought," deny the impoverishment of auriferous lodes in depth.§

* *A Treatise on Ore-Deposits*, by J. A. Phillips, London, 1884, pp. 60–62.

† *Gold-Fields of Victoria*, Melbourne, 1869, pp. 233-281.

‡ *The Gold-Fields of Victoria in* 1862, by a Special Reporter of the *Argus*, Melbourne, 1863.

§ The gold-field of Bendigo is often cited as an instance to the contrary, good yields having been there obtained in some cases from great depths (2000 to 2800

This greater richness of the vadose region might be explained in either of two ways.

1. If the auriferous contents of the lodes are derived from some deeper source, and have been deposited from warm ascending waters, the decrease of pressure on approaching what was, at the time of the lode-formation, the surface, might account for the precipitation of the precious metals in greater quantities near that surface.

2. If the reefs and their auriferous contents are due to the leaching action of solutions traversing the country-rock on each side of the fissures, such leaching action would naturally be far more intense near the surface, because the oxidizing action of the surface-water would naturally be much greater in the vadose region.

Either of these hypotheses might account for the greater richness of the vadose region. In the first case we might have in the present deposits of that region an example of the re-solution and precipitation of gold which had been previously brought up from greater depths; while in the second case we might expect to trace a leaching action similar to that occurring in depth, only augmented by the effect of the oxidizing vadose waters.

The question is naturally suggested, whether natural reagents capable of dissolving gold* are to be found in the vicinity of auriferous lodes. And this inquiry suggests the further questions: Does gold exist in solution in mine-waters of either the vadose or the deep circulation? Does the gold of the vadose circulation, in any particular mine, contain a smaller percentage of silver than that of the deep circulation? Is any evidence of re-solution and re-precipitation to be obtained by analyzing samples of country-rock from the vadose region, corresponding in position with samples of the deep region of the same district?

Natural Solvents of Gold.

The chief solvents of gold at all likely to occur in the neighborhood of lodes are bromine, iodine, ferric chloride, ferric sulphate and chlorine.

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feet); but the reason for this is indicated in the remarks of Mr. E. J. Dunn, in his report on the *Bendigo Gold-Fields*, pp. 6 and 9.

* Gold is so easily precipitated from solution that an investigation as to possible precipitating agents is scarcely necessary.

Bromine and Iodine.—The action of iodine as a solvent for gold in nature has been emphasized by some writers;* but, whatever may have taken place in the past, the present investigation seems to show that iodine is not at all abundant in auriferous rocks. The partial or complete analysis of 53 samples of mine-water from both the vadose and the deep circulation has detected no bromine, and only in one instance any trace of a soluble iodide. If, however, bromides and iodides do exist in the vadose region, the agents which liberate chlorine (considered below) would also liberate bromine and iodine.

Ferric Chloride and Ferric Sulphate.—Mr. Henry Wurtz† remarks that as early as 1859 he called attention to the solubility of gold in these salts; but he does not state the strength of the solvent solutions employed. The same is true of many other assertions of this reaction, encountered in technical literature.‡ It was therefore deemed necessary in the present investigation to test the solubility of gold in solutions of the above salts of various strengths, not greatly exceeding, however, the degree of concentration actually found in the most highly mineralized waters analyzed. Solutions of ferric chloride and ferric sulphate, containing from 1 to 20 grammes per liter, were prepared, and finely divided (1) metallic gold and (2) auriferous sulphides were treated in these solutions, being freely exposed to the air at ordinary temperature for several months; but no gold was dissolved, even by the strongest solutions. As the highest strength above named considerably exceeded that of the most highly mineralized mine-waters analyzed, no experiments were made with still stronger solutions of the ferric salts.

The negative result of these experiments is of course not conclusive proof that gold may not have been dissolved by these reagents in the vadose circulation in a longer time and under other conditions than those supplied.

Chlorine.—Possibly many reactions in nature, not easily reproduced in the laboratory, may liberate chlorine, even at ordi-

* For example, T. A. Rickard, "On the Origin of the Gold-Bearing Quarts of the Bendigo Reefs." *Trans.*, xxii., 309.

† "Gold-Genesis," *Scientific American Supplement*, No. 977, p. 15,644.

‡ It is highly desirable that in all such statements of solubility, the precise strength of the reagents should be given. Most of the accounts of the solution of gold, for instance, employ, at best, only the vague terms "strong," "dilute," etc.

nary temperatures. We know, however, that it is produced by the action of hydrochloric acid on the higher oxides of manganese, or by the action of sulphuric acid on the same oxides in the presence of chlorides.

The question whether agents for the re-solution of gold exist in the vadose region is thus practically narrowed to a search, in the waters and rocks of that region, for (1) free hydrochloric acid; (2) free sulphuric acid; (3) the higher oxides of manganese; and (4) ferric chloride and ferric sulphate.

It was desirable, at the outset, to determine the most dilute solution of hydrochloric acid which will, in the presence of the higher oxides of manganese, liberate sufficient chlorine to be detected by ordinary tests. Experiment showed that one part of hydrochloric acid of 1.16 sp. gr. in 2500 of water would give a distinct chlorine reaction, while one part of the same acid in 1250 of water produced chlorine enough to dissolve an amount of gold appreciable by delicate tests. As the proportion of pure HCl to water is in the first case only about 1 to 8000, and in the second case 1 to 4000, it is evident that extremely dilute acid will, in the presence of manganese oxides, dissolve gold.

Cause of Acidity in Mine-Waters.—The chief cause of acidity in mine-waters (see examples below) is without doubt the oxidation of pyrite, which yields ferric sulphate and sulphuric acid. The latter, acting on the chlorides, which are always present to greater or less extent in mine-waters, frees hydrochloric acid. The writer has never found a water containing free acid in which there was not also a large percentage of ferric salts.

The Occurrence of Oxides of Manganese in Mining Districts.—In some mining districts (notably in Karangahake, in the Thames gold-field) the oxides of manganese often form a great part of the lode-filling. While this, however, is exceptional in Australia and New Zealand, the presence of the higher oxides of manganese in the ferric oxides of the vadose circulation is surprisingly general. Twenty analyses of such material from various localities showed in 17 cases manganese, representing from 0.012 to 43.59 per cent. (reckoned as Mn_3O_4). To one sample, containing only 0.38 per cent. of Mn_3O_4, dilute hydrochloric acid and precipitated gold were added, and gold was found to be dissolved.

If, therefore, the vadose mine-waters are found to contain free hydrochloric acid, it is evident that agents for the re-solution of gold in that zone are not lacking.

The Acidity of Vadose Mine-Waters.—An acid reaction with test-paper does not prove the presence of free acid. Every water examined which contained an appreciable quantity of ferric salts gave a distinct acid reaction, though in a number of cases examination proved the absence of free acid.

Seventeen samples of vadose waters were examined for free acid; care being taken to collect the water as it ran from the rock or vein, before any considerable exposure to oxidizing agencies other than the oxygen held in solution by the water itself.

In calculating the results from those samples which carried much free acid, if both sulphates and chlorides were present, and the amount of free acid exceeded the amount represented by the chlorine radical in the water, the whole of the chlorine radical was taken as combined with H to form free hydrochloric acid, and the remainder of the free acid found was reckoned as sulphuric acid. The results are shown in Table XIX. The amount of ferric chloride and sulphate can be approximately calculated from the proportion of iron present as ferric salts. Even after complete oxidation by exposure to the air, the total weight of ferric salts could never exceed 12 grammes per liter. For this reason, in the experiments previously described. (*ante*, p. 598), I did not use solutions of ferric salts containing more than 20 grammes per liter.

Table XIX. shows the considerable increase in acidity caused by exposure to the air. It is noteworthy that all the samples marked *, when taken from the mine, precipitated gold from solution, but that the same waters, after thorough oxidation, dissolved metallic gold when the higher oxides of manganese were added to them.

The results shown in Table XIX. point to the following conclusions:

1. In districts like the Thames, N. Z., where the country-rock is highly charged with sulphides, the vadose water may often contain free hydrochloric acid sufficient (when the higher oxides of manganese are present) to re-dissolve gold. Though the Thames samples were incapable of holding ordinary salts

of gold in solution, they acted as solvents of gold when they were thoroughly oxidized and manganese oxides were present.

2. The great majority of the mine-waters analyzed contained no free acid which could liberate chlorine by acting on the oxides of manganese that are abundant near quartz reefs.

3. The higher salts of iron are not present in any samples of water analyzed by me, in sufficient quantity to dissolve gold at ordinary temperatures. (Stronger solutions of these salts failed to dissolve gold.) It may be added, that in every case in which much iron was present, free acids were also found; so that in any solution of gold that might be effected, the more powerful solvent, chlorine, might also be acting.

Notwithstanding these conclusions, I must point out that the re-solution of gold has probably gone on, and is still going on, in the vadose region, even where the vadose waters contain neither free hydrochloric acid nor notable quantities of ferric salts. The analyses of samples from the vadose regions of Walhalla and Ballarat (see Tables XX. and XXI., and Diagrams 8 and 9) the vadose waters of which contained no free acid, and were very poor in dissolved minerals, show that such re-solution has probably been considerable, though we find no agencies now existing which would account for it.

Does Gold Exist in Mine-Waters of Either or Both Circulations?

Prof. A. Liversidge* has pointed out that the search for gold in meteoric and mine-waters has not proved its presence in solution. It has been detected, but it may have been in mechanical suspension. So far as I know, Messrs. Norman Taylor and Cosmo Newbery, of the Victorian Geological Survey, are the only persons who have experimentally investigated this subject in these colonies. Mr. Newbery, who made the most experiments, said before the Victorian Royal Commission on Gold-Mining† that whenever he got gold, he got also angular fragments of quartz, which could find its way wherever gold could find its way, and both might have been conveyed mechanically.

The evidence for the existence of gold in mine-waters rests, so far as I am aware, on the discovery of gold: (1) in boiler-

* "On the Origin of Gold Nuggets." *Roy. Soc. N. S. Wales*, Apr. 6, 1893.

† *Report of the Commission*, 1893, p. 68.

scale from boilers fed with mine-water; and (2) in wood taken from old mine-workings, where it had been covered for some time with mine-water—the latter being assumed to have carried dissolved gold into the timber, to be precipitated by the organic matter of the wood. But the finding of gold under such circumstances does not prove that it was in solution in mine-waters at deep levels. In the first case, the gold may have been carried into the boiler in suspension, along with the silt which all mine-waters contain. In the second case, even though gold may have been dissolved in the water surrounding the old timbers, it may have been brought into such solution by the action of air in the mine-workings, oxidizing sulphides of the rock to sulphates and setting free sulphuric acid, which, in turn, acting on the chlorides always present in mine-waters, would liberate hydrochloric acid. This acid, acting on oxides of manganese, would free chlorine, which would dissolve gold. This statement applies particularly to all mines the waters of which contain much iron. Every sample of chalybeate mine-water analyzed by me acted as a precipitant of gold when taken fresh from the workings, but as a solvent of gold at ordinary temperatures, in the presence of the oxides of manganese, when it had been exposed to the air for a week or two.

Mr. Newbery, however, distinctly said, in his testimony already cited, that he found angular quartz which had been soaked up into the timber examined, and that the gold might have been mechanically introduced in the same way.

With regard to the suspension of gold in mine-waters, the following evidence, obtained by me last year, may be of interest.

In the Long Tunnel G. M. Co.'s mine at Walhalla, Gippsland, Victoria, one of the most productive mines in Australia,* the water pumped from various depths, down to about 2300 feet below the surface, is run into two large settling-tanks, before using. At the time of my visit one of these tanks contained a large quantity of fine silt, which had been suspended in the mine-water. I analyzed three samples of about 2 pounds each, first panning off the lighter part, and then assaying the residue. The first sample gave 0.0063 grain of gold; the second, no trace; and the third, 0.0175 grain.

* Mr. Ramsay Thompson, the general manager, to whom I am indebted for much kind assistance, informed me that up to December, 1894, this mine had produced over 22 tons of gold, and had paid £1,200,000 in dividends.

These results show that assays of boiler-scale do not necessarily prove that gold was dissolved in the water depositing it; and, also, that in all analyses even of samples from the vadose circulation, to test the presence of dissolved gold, care must be taken to free the water beforehand from every trace of suspended matter.

I have tested many old mine-timbers for gold. In every case the outside wood was chipped off to the depth of about ½ inch; and, when cracks appeared in the timber, about ½ inch on each side of the crack was also chipped off. These parts were burned, and analyzed separately from the inner portions.

The following table shows the results of several such analyses.

Analyses of Old Mine-Timbers.

Sample.	Description and Locality.	Part Analyzed.	Gold Found.
I.	Prop from the Tipperary Gold Mining Co., Macetown, Otago, much decomposed. The part analyzed had been under water for many years.	Outside of the prop.	None.
II.	Do.	Inside of do.	None.
III.	Prop from an old tunnel in an alluvial terrace, near Skipper's Point, Otago. The part analyzed had been under water for probably 20 years.	Outside of prop.	None.
IV.	Do.	Inside of do.	None.
V.	A prop from the Premier mine, Macetown, Otago. Water had been running over it for 2 years at least.	Outside of prop.	0.0038 grain.
VI.	Rotten wood from the lower part of a rejected prop from the Long Tunnel Gold Mining Co., Walhalla. From a part of the mine where a richly auriferous reef had been worked.	Outside of do.	0.0097 grain.
VII.	Do.	Inside of do.	None.
VIII.	Decayed portion of lath brought up from the deep workings of the Northern Star Gold Mining Co., Ballarat. Had evidently been saturated with water for many years.	No portion was sound; all was burned and analyzed.	None.
IX.	Portion of sleeper used in a tramway, Northern Star Gold Mining Co., much decayed. A large quantity of water had been running over it for at least a year.	No portion was sound; all was burned and analyzed.	0.0129 grain.

Search for Gold in Mine-Waters.—It was soon found useless to examine mine-waters which contained much iron, and in which the ferrous salts had not been oxidixed to ferric salts by exposure to the atmosphere. The analyses were consequently restricted, for the vadose region, to waters containing a very small percentage of iron salts, and chalybeate waters which have been thoroughly oxidized (the latter being apparently exceptional, even in that region), and, for the deep circulation, to waters containing so little iron that they do not act as precipitants of gold.

This chapter treats of mine-waters under the conditions of temperature and pressure now encountered. What might be effected by these waters under other conditions does not concern us at this stage.

If gold be present in mine-waters at all, it is likely to be in very minute proportions. Hence large quantities of water must be operated upon. The method of evaporation for the assay of the residue was too tedious, especially in view of the limited time at my disposal in each locality. I therefore availed myself of the well-known action of sulphides and organic matter in precipitating gold from solution.

A filter was constructed, consisting of a tinned-iron cylinder, about 3 inches in diameter and 6 inches long, terminating below in a funnel, inside of which was placed a filter of glass-wool, and above this the reducing agents (animal charcoal, artificial iron and lead sulphides, roughly powdered). The upper part was connected by a rubber tube with the tap supplying the water to be tested.

The following preliminary test proved the efficiency of this filter: A solution in 400 gallons of water (from the Dunedin water-mains) of 28 pounds of common salt, 8 ounces of magnesium sulphate, 8 ounces of ferric chloride and 0.1 grain of gold in the form of auric chloride, was allowed to trickle slowly through the filter, the operation taking about 48 hours. The mixture of sulphides and charcoal was then removed, roasted and assayed, when 0.0926 grain of gold was recovered, showing that practically all the gold had been precipitated.

In examining mine-waters, the water was first collected in an iron tank, and powdered alum was added to it, completely precipitating all suspended matter. After standing some hours,

it was tapped from 2 or 3 inches above the bottom of the tank, and filtered as above described.

Four samples were thus treated; but in no case was gold found.

The first sample (about 500 gallons) was water from the high-level tunnel of the Tipperary G. M. Co., which had percolated down from the surface through the lode-fissure for from 200 to 500 feet, and flowed in a large stream from the tunnel-mouth. This practically represented the vadose circulation only.

The second sample (about 500 gallons) was from the pump-discharge of the New Chum Railway Co., Bendigo, Victoria, and represented the whole drainage of the mine from the surface to a depth of 2850 feet.

The third sample (about 700 gallons) was from the Long Tunnel G. M. Co.'s mine, Walhalla, Victoria, near a very rich lode, and represented the whole drainage between the adit (700 feet below the summit of the hill) and a level about 2300 feet below the surface, or over 1500 feet below the adit.

The fourth sample (about 500 gallons), from a deep tunnel of the Premier mine, Advance Peak, Otago, driven on a lode which had proved richly auriferous in places, represented the whole drainage from the surface to probably 2000 feet below it.

The negative results of these tests are the more surprising to me, since other examinations, hereinafter described, afforded strong evidence that solution and re-precipitation of gold have taken place in the vadose region.

It is, of course, possible that gold may have existed in these samples in some form from which it was not precipitated by the reagents used. This is suggested, indeed, by my experience (see Chapter VII.) in attempting to precipitate gold from sea-water. The question can only be decided by the evaporation of samples (first freed from suspended matter) in larger amount and number than mine, and the assay of the residues. I venture to recommend such an inquiry to those who live in the vicinity of rich mines and have time for the work.

Does the Gold of the Vadose Region Contain Generally Less Silver than That of the Deep Circulation in the Same District?

Several investigations have seemed to prove that, on the average, the gold of the alluvial deposits in Australia and elsewhere

is appreciably finer in quality than the vein-gold. The matter has been discussed chiefly in connection with the origin of nuggets;* but, so far as I know, no comparison as to fineness has been made between the vadose and the deep vein-gold.

The observed difference between placer- and vein-gold may be held to show, either that the oxidized mineral-bearing waters running in the ancient drift-deposits dissolved out part of the silver with which the gold was alloyed, or else that these waters dissolved both gold and silver, the gold being again precipitated, alloyed with less silver than before. Similar reasoning might be applied to an observed difference between the vadose and the deep zone, in the quality of vein-gold. I have therefore made comparative assays of this character in a number of cases.

The fineness of the gold may vary considerably even in the same level, and within a few feet. This is true even in such districts as Bendigo, Ballarat and Otago, where the percentage of silver is comparatively low, while in districts like the Thames, N. Z., the variation observed is sometimes extraordinary. On the whole, however, the average quality of the gold won in districts of the former class varies little.

Five localities were chosen (chiefly by reason of facilities for obtaining specimens from near the surface), namely, (1) the Nenthorn gold-field, in mica-schist, in eastern Otago; (2) the Tipperary and Premier mines, in mica-schist, Macetown, central Otago; (3) the Dart river, in northern Gippsland, representing the Upper Silurian of Victoria; (4) the Bendigo field,† representing the Lower Silurian; and (5) the Thames district of the North Island, in altered Lower Tertiary andesite.

In a few instances the analyses were made of gold picked out of the reef; but the majority were assays of vein-stone. The percentage of silver was in each case obtained by difference, and represents the loss per cent. of the prill after quartation.

* Prof. Liversidge, in the paper already cited, gives a convenient summary of the literature of this subject.

† Surface samples are hard to get in Bendigo. Nearly all the companies are mining in the deep region, and have long ago exhausted the pay-quartz above. I took many samples from outcrops of reefs, but found in the majority little or no gold. The results given below are those in which a prill of appreciable size was obtained by assaying 1000 grains of vein-stone.

The results were as follows:

Relative Fineness of Vadose and Deep Vein-Gold.

A. Surface Samples from Nenthorn Gold-Field, Otago, N. Z.

No.	Percentage of Gold.	Percentage of Silver.
1	91.85	8.15
2	89.27	10.73
3	90.43	9.57
4	89.19	10.81
5	92.01	7.99
6	89.38	10.62
7	91.46	8.54
8	90.79	9.21
9	90.07	9.93
10	92.00	8.00

Average fineness of 10 samples, 90.645 per cent. Range of fineness, 89.19 to 92.01 = 2.82 per cent.

B. From Deeper Levels of Nenthorn District (the Bulk of the Gold was Held by Pyrite).

No.	Percentage of Gold.	Percentage of Silver.
1	89.86	10.14
2	91.43	8.57
3	87.17	12.83
4	90.45	9.55
5	89.07	10.93
6	92.01	7.99
7	90.17	9.83
8	88.75	11.25
9	91.03	8.97
10	90.30	9.70

Average fineness of 10 samples, 90.024 per cent. Range of fineness, 87.17 to 92.01 = 4.84 per cent.

C. Surface Samples from Tipperary, Premier and Sunrise Gold Mining Companies. All near Macetown, Central Otago, N. Z.

No.	Percentage of Gold.	Percentage of Silver.
1	96.12	3.88
2	93.37	6.63
3	94.19	5.81
4	92.87	7.13
5	95.00	5.00
6	94.58	5.42
7	93.23	6.77
8	95.84	4.16
9	95.93	4.07
10	94.89	5.11

Average fineness of 10 samples, 94.602 per cent. Range of fineness, 92.87 to 96.12 = 3.25 per cent.

D. From Deep Levels (1000 to 1500 Feet Below Surface) in Tipperary and Premier Mines, Macetown.

No.	Percentage of Gold.	Percentage of Silver.
1	95.39	4.61
2	94.48	5.52
3	91.87	8.13
4	96.29	3.71
5	93.06	6.94
6	94.31	5.69
7	93.80	6.20
8	95.73	4.27
9	93.90	6.10
10	94.10	5.90

Average fineness of 10 samples, 94.293 per cent. Range of fineness, 91.87 to 96.29 = 4.42 per cent.

NOTE.—A sample taken from a bar of 194 ounces from deep levels of Premier Gold Mining Co. contained 94.45 per cent.

These analyses of 76 samples go, on the whole, to prove that the average fineness of the gold in the vadose region is appre-

E. Surface Specimens from Richly Auriferous Lode in the Vicinity of the Dart River, North Gippsland, Victoria.

No.	Percentage of Gold.	Percentage of Silver.
1	91.87	8.13
2	94.96	5.04
3	93.08	6.92
4	92.00	8.00
5	92.63	7.37
6	94.73	5.27
7	93.45	6.55
8	95.01	4.99
9	94.91	5.09
10	92.84	7.18

Average fineness of 10 samples, 93.548 per cent. Range of fineness, 91.87 to 95.01 = 4.14 per cent.

F. Unoxidized Specimens from the Same Reef, Dart River, North Gippsland, Victoria.

No.	Percentage of Gold.	Percentage of Silver.
1	91.50	8.50
2	90.43	9.57
3	87.69	12.31
4 Note (a)	96.28	3.72
5	91.17	8.83
6 Note (b)	84.64	15.36
7	88.93	11.07
8	95.03	4.97
9	90.37	9.63
10	90.80	9.20

Average fineness of 10 samples, 90.984 per cent. Range of fineness, 84.64 to 96.28 = 11.64 per cent.

Note (a).—Sample assayed consisted of arsenopyrite and pyrite.

Note (b).—Sample assayed composed chiefly of galena.

G. Surface Samples from Bendigo Gold-Field, Victoria.

No.	Percentage of Gold.	Percentage of Silver.
1	94.76	5.24
2	95.03	4.97
3	94.05	5.95
4	94.27	5.73
5	94.10	5.90
6	95.00	5.00

Average fineness of 6 samples, 94.535 per cent. Range of fineness, 94.05 to 95.03 = .98 per cent.

H. Samples from Deep Levels (1200 Feet to 3000 Feet) of Bendigo Gold-Field.

No.	Percentage of Gold.	Percentage of Silver.
1	93.08	6.92
2	95.04	4.96
3	94.00	6.00
4	91.27	8.73
5	94.30	5.70
6	94.19	5.81
7	93.96	6.04
8	92.95	7.05
9	92.37	7.63
10	94.70	5.30

Average fineness of 10 samples, 93.586 per cent. Range of fineness, 91.27 to 95.04 = 3.74 per cent.

ciably greater than in the deep circulation in the same district, and also that the vadose gold is considerably more regular in quality.

It seems to be indicated that considerable solvent action must have been exercised by water percolating through the rocks of

I. Samples from Vadose Region of Various Parts of Thames District, N. Island, N. Z.			K. Samples from Deep Circulation of the Thames District.		
No.	Percentage of Gold.	Percentage of Silver.	No.	Percentage of Gold.	Percentage of Silver.
1	63.72	36.28	1	63.76	36.24
2	79.34	20.66	2	71.14	28.86
3	52.90	47.10	3	59.93	40.07
4	57.39	42.61	4	53.01	46.99
5	67.01	32.99	5	64.25	35.75
6	59.98	40.02	6	43.17	56.83
7	65.55	34.45	7	66.98	33.02
8	73.07	36.93	8	51.30	48.70
9	60.08	39.92	9	69.04	30.96
10	51.19	48.81	10	48.75	51.25
Average fineness of 10 samples, 63.023 per cent Range of fineness, 51.19 to 79.34 = 28.15 per cent.			Average fineness of 10 samples, 59.133 per cent. Range of fineness, 43.17 to 71.14 = 27.97 per cent.		

the vadose region, as the denudation of the surface has gradually lowered the water-level, converting the deep circulation of former times into the vadose of to-day.

Analyses of Vadose Country-Rock, etc., at Different Distances from Auriferous Lodes.

At the beginning of this chapter I have pointed out the importance of separate rock-assays in the vadose region. It was relatively difficult to obtain good samples (other than surface-samples) of this class, because most of the mines are now deep, and the long cross-cuts formerly run on upper levels into the country-rock are abandoned and closed. The samples therefore comprise chiefly oxidized rock from pretty near the lodes, and 10 to 100 feet below the surface, and surface-samples taken at all distances from the lodes. Particular interest attaches to samples of (1) oxide of iron and manganese deposited along bedding-planes or fractures; (2) solid rock as little altered as possible by the action of percolating water; and (3) secondary* sulphides, abundant in the vadose region of many mines.

* In calling these vadose sulphides "secondary," I mean that they have probably been formed through the oxidization of the sulphides of the deep circulation by surface-water, followed by a reduction of the sulphates and re-precipitation of the sulphides by organic matter. I do not mean to say positively that the sulphides of the deep zone are not, as many observers believe, also secondary in this

The remarks on p. 565, covering the methods of concentration and assay pursued with samples from the deep levels, apply here also. But concentration of an oxidized rock is much more difficult, because oxidation destroys the heavy sulphides, and also liberates very finely divided gold, which there is danger of losing. Hence my results with vadose country-rock are not quantitatively correct. To minimize the probable error, the samples were not concentrated nearly as far as those from deep levels had been.

Walhalla.—A good cross-section of the gold-bearing rocks of Walhalla is exposed along both sides of the Walhalla creek, where the cliffs rise steeply from 300 to 600 feet. Analyses of specimens are given in Table XX.

Ballarat.—A deep cutting, about ¼-mile long, running E., across the strike of the rocks, from the summit of Sovereign hill, in this district, presented a good cross-section of the country-rock near the surface. Table XXI. gives the result of analyses.

Otago.—Examinations of country-rock from the deep region in Otago were confined to one district, Macetown; but quartz-reefs have been worked at various depths in different parts of the province; and vadose samples were taken from three other Otago districts besides Macetown, namely, Waipori, Nenthorn and Saddle Hill. In every case the country-rock is either phyllite or mica-schist. Table XXII. gives the analyses of 13 samples.

Otago.—Samples from Districts Remote from Auriferous Lodes.—Vadose samples were also taken in Otago, far from any auriferous lodes. Previous assays having shown that the materials most likely to contain gold were the broken rock, iron oxide, etc., filling fault-fissures, the samples were taken of such materials only. If all the mica-schists of the Otago gold-field contained gold, some of it would be carried by percolating water into such crevices, and lodge there with the ferric oxide. The analysis of samples at a long distance from any auriferous reef is specially interesting, since, as will be seen in the last three

sense, that is, due to the reducing action of carbonaceous matter upon soluble sulphates. This is Sandberger's view (*Untersuchung*, etc., Vol. I., p. 21). Yet so far as my experience of the deep sulphides goes, it certainly favors the theory of their formation by the action on the silicates of metals of hydrogen sulphide, dissolved in ascending water.

tables, deposits of ferric oxide in the vadose region, even at a *considerable* distance from a reef, were nearly always auriferous.

A good section of the favorable rock (the middle division of the foliated schists already mentioned), in which, however, for several miles, no gold-bearing reef has yet been discovered, is exposed by the "Skipper's" road from Queenstown to Skipper's creek. This road is cut round almost vertical cliffs for several miles, on the north side of the Skipper's range. Nine samples of broken rock and ferric oxide from joints, cracks and faulted country in these cuttings were analyzed with the results shown in Table XXIII.

Lake Wakatipu.—Fourteen vadose samples of little-altered Upper Devonian and Carboniferous rocks were taken from the western shore of Lake Wakatipu. No gold was found in them. The examination is reported later on, under the head of "Gold in Marine Sediments" (p. 615).

Ohinemuri District, Thames.—The samples collected for me from the vadose region in the Ohinemuri district, in the southern part of the Thames gold-field, differ somewhat from the oxidized samples of other gold-fields. All those analyzed are highly altered andesites; and, in many cases, even when most oxidized, they contain much pyrite. This sometimes doubtless represents the pyrite found in the propylites of the deep region in this district; but, from the mode of its occurrence, I am inclined to think the greater part of it is due to the oxidation of the older pyrite to ferrous and ferric sulphate, and the subsequent reduction of such sulphates by organic matter. I have therefore called it "secondary" pyrite. (The bullion associated with it does not carry the abnormally high percentage of silver which was noticed in the bullion from the sulphides of the deep region—a fact which should repay further investigation, and might throw important light upon the solution and re-precipitation of gold and silver by natural agents.) Table XXIV. gives the results of the analyses of these samples.

Remarks.—These examinations show a striking difference in gold-contents between the vadose and the deep region of the same district. Diagrams 8 and 9 show this difference graphically, in curves plotted for the vadose samples and for samples from the deep region (900-foot and 1422-foot level) of the Walhalla Long Tunnel mine.

In the deep region, as has been shown, gold was obtained only when pyrite was present in the rock; and when such pyrite occurred at a considerable distance from the reef, it was seldom gold-bearing. In the vadose region, on the contrary, the country-rock was found to be impregnated with gold to a much greater distance from the reef, and to a much greater degree.

It is of course possible that some of the gold found in the vadose country-rock was carried into it mechanically by percolating surface-water, and its presence may therefore be no proof of the solution and re-precipitation of gold. But in view of the positions from which most of the samples were taken, I think the results indicate that such solution and re-precipitation have gone on to a considerable extent in the vadose region—the gold being in all probability derived from higher parts of the lode, which have long since disappeared through surface-detrition.

Chapter VII.—The Origin of Gold in Stratified Deposits.

The country-rocks of nearly all the chief Australian gold-fields are more or less altered sedimentaries, originally deposited in marine basins. Hence writers on the origin of the gold in the reefs have laid much stress on the presence of minute quantities of gold and silver in sea-water. The argument is briefly:

Gold exists in sea-water. Paleozoic marine sediments therefore contained gold, either mechanically entangled in them, or precipitated with them by organic matter, which undoubtedly existed in the ancient seas. These horizontal deposits being subsequently tilted and fractured, their gold and silver were re-dissolved by percolating waters and re-precipitated in the lode-fissures where they are now found.

This has been the thesis of not a few ingenious speculations, backed sometimes by chemical equations, but not by chemical analyses.

In 1851, Malaguti and Durocher announced the discovery of silver in sea-water, and made a quantitative estimation of it, namely, 1 milligramme in 100 liters (or 0.155 grain per ton). But they did not report any gold.

In 1872, E. Sonstadt* discovered gold in sea-water from Ramsey Bay, on the coast of the Isle of Man. He did not determine the quantity, but said it was certainly less than 1 grain per ton. Strange to say, many writers who have used this discovery as a basis for theoretical speculation have represented Sonstadt as having found 1 grain per ton.†

So far as I am aware, no attempt has been made to verify Sonstadt's discovery, and to determine accurately the amount of gold in sea-water, or to test his statement that this gold is not precipitated by ordinary reducing-agents, by reason of the presence in sea-water of iodate of calcium.‡

Methods of Detecting Gold in Sea-Water.

Sonstadt's Methods.—Sonstadt gave three methods for the detection of gold in sea-water, two of which he recommended as easily applicable. In the first of these, 150 to 200 c.c. of sea-water is acidulated with hydrochloric acid, ferrous sulphate is added, and the water is concentrated by boiling. The film of ferric oxide found in the bottom of the dish is treated with chlorine-water, and the solution obtained is tested for gold with tin chloride. In the second method, a small quantity of baric chloride is added to the sea-water, and both gold and silver are found to be precipitated.

The first method I tried upon water from the Pacific, at St. Clair head, near Dunedin, and obtained with tin chloride a distinct coloration, doubtless due to gold.§ But I found it impossible to obtain beads of gold by fusing with borax and pure lead, as Sonstadt directs. (This is not surprising, since my

* "On the Presence of Gold in Sea-Water."—*Chem. News*, October 4, 1872, vol. xxvi., p. 159.

† Thus Mr. James Park in his report on the Thames Gold-Field, already cited, says (p. 65): "Sonstadt was the first to show that every ton of sea-water contains a grain of gold." A later writer, noting, perhaps, that Sonstadt found "less than a grain," is very scrupulous, and fixes the amount at 0.9 grain!

‡ Since my experiments were made, I have learned from Mr. Rickard's paper on "The Origin of the Gold-Bearing Quartz of the Bendigo Reefs" (*Trans.*, xxii., 308) that Münster has found gold in the sea-water of Christiania fiord, and has estimated the amount. I find also that Prof. A. Liversidge has estimated that the sea-water off the coast of N. S. Wales contains 0.5 grain of gold to the ton. (See his paper, read before the N. S. W. Royal Society, October 2, 1895, "On the Amount of Gold and Silver in Sea-Water.")

§ The experiment is more successful with the modification of the tin-chloride test prepared by T. K. Rose (*Chem. News*, Dec. 2, 1892, vol. lxvi, p. 271).

subsequent determinations would give, as the quantity of gold in 200 c.c. of this sea-water. less than 0.000012 grain.) The necessity of boiling makes this method unsuitable for the treatment of samples large enough to yield a weighable bead of gold.

I therefore tried the precipitation with baric chloride. Sonstadt makes the remarkable assertion that in order to precipitate the gold, the baric chloride does not need to be added in sufficient quantity to precipitate as baric sulphate all the soluble sulphates in the sea-water, but, on the contrary, that the amount added to a liter of sea-water need not exceed that required to form about 1 grain of precipitate. Baric sulphate being one of the least soluble of salts, this statement seems inexplicable; nevertheless, I have been convinced, whatever be the explanation, that Sonstadt's method is as effective as if the whole of the soluble sulphates were precipitated as baric sulphate.

He explains the precipitation by baric chloride by supposing the gold to be present as an aurate. To test this question, artificial sea-water was prepared, and the aurate of potassium was added to it. Subsequent treatment with baric chloride precipitated no gold. The experiment was repeated, with the same result.

It seems unlikely, therefore, that the gold exists in sea-water as an aurate. I confess that I can form no conception of its state of combination. The subject would repay a more thorough investigation.

The Author's Method.—Having been led, by evidence which I will not here repeat, to doubt whether the precipitation with baric chloride was complete, I tried a different one, which, if successful at all, would certainly precipitate all the gold present. All ordinary salts of gold are reduced to the metallic state by moderate heating. Applying this principle first to an artificial solution, I added to 112 pounds of artificial sea-water* ten times as much calcic iodate as Sonstadt found, and a solution of auric chloride in sodium chloride, containing 0.005 grain of gold, so that the water would contain 0.1 grain of gold per ton. This solution was allowed to stand in a dark place for a week, that

* Prepared according to the analysis by Roscoe and Schorlemmer.

the gold might have time to form possible new combinations. The whole was then evaporated; the mixed salts (over 4 pounds) were heated dull red and lixiviated with water; the insoluble residue (123 grains) was fused with borax and pure litharge, and 0.0043 grains of gold was obtained. The experiment was repeated, the gold being added as an iodate, dissolved in excess of potassic iodate; and in this case 0.0052 grain of gold was recovered. (The slight excess may have been due to a small particle of the cupel remaining in the button. It was not due to silver in the litharge.)

The method was then applied to actual sea-water; only the sample was doubled in weight, to allow for the smaller proportion of gold. The mixed salts (about 8 pounds) resulting from the evaporation of 0.1 ton of sea-water were heated dull red and lixiviated, and the residue (principally sand, with a little oxide of iron) was fused and cupelled as before. The weight of pure gold obtained from 0.1 ton of sea-water was, in the first experiment 0.0065, and in the second 0.0071 grain. In both cases, the prill contained absolutely no silver.

The following table summarizes the experiments above described:

Determinations of Gold in Sea-Water.

Method.	Sea-Water.	Gold Obtained.	Gold per Ton.
1. Sonstadt's (baric chloride):	Pounds.	Grains.	Grains.
a. All sulphates precipitated	224	0.0061	0.061
	112	0.0038	0.076
b. Small proportion of sulphates precipitated	224	0.0074	0.074
	224	0.0078	0.078
2. Evaporation and reduction of gold in residue by heating	224	0.0065	0.065
	224	0.0071	0.071
Average per ton	.....	.....	0.071

I have said nothing concerning the silver in sea-water. Precipitation by baric chloride certainly saves some silver, but only about one-fourth as much as was reported by Malaguti and Durocher. Probably the precipitation is not complete.

The Precipitation of Gold in Marine Sediments.

Since gold exists in sea-water, it seems reasonable to believe that it is precipitated at the present time by natural reducing-

agents. Those writers who trace the metallic contents of lodes to metals dissolved in sea-water assume, indeed, that such a precipitation is constantly going on; but experimental proof of this assumption is lacking. I have attempted to investigate the question in two ways:

1. By the examination of coast-sediments, now being deposited under conditions favorable to the reduction of gold from the sea-water.

2. By the introduction into sea-water of reducing-agents such as naturally occur along the coasts at the present time, and a subsequent examination for precipitated gold.

Examination of Coast-Sediments.—To secure trustworthy results, the whole of the drainage-basin to the erosion of which the sediments are due should consist of non-auriferous rocks, so that we may be sure that any gold detected did not come from the land. Otago Harbor, on the upper part of which Dunedin is located, satisfies this requirement. The Leith and other small streams entering the harbor flow wholly through basic Tertiary volcanic rocks which contain no gold. At the same time the shores are more or less covered with timber, so that organic matter is abundant in the sediments of the streams. The volcanic rocks contain much iron. Beds of hematite and limonite abound along the shores. The conditions for the reduction of gold from sea-water are therefore very favorable. Besides organic matter, there is sulphide of iron, produced by the action of the sulphates in sea-water upon iron salts, forming sulphate of iron, reducible to sulphide by organic matter. (As will be seen, some such action does in fact take place. In every case the concentrates contained insoluble sulphides.)

Analyses were made of mud and silt from different parts of Otago Harbor, where the circumstances seemed most favorable. In each case, from 1 to 2 hundredweight of the silt was carefully panned off, till a residue of about 1000 grains was left. This residue consisted chiefly of magnetite, augite and hornblende, derived from the volcanic rocks of the coast. The percentage of sulphur (insoluble sulphides) in the concentrate was determined upon a small portion, and the remainder was roasted (giving in every case a strong reaction of sulphur dioxide).*

* When the magnetite was removed by means of a magnet, the remainder did not show, under the microscope recognizable crystals of pyrite; but the presence of pyrite was certainly proved by roasting, and by the sulphur-determination.

The roasted material was then assayed. The details of four assays are given in the following table:

Examination of Marine Sediments.

Sample.	Concentrates.		Sulphur in Non-Magnetic Concentrates. Per cent.	Gold Obtained.
	Total Grains.	Extracted by Magnet. Grains.		
a	438	365	12.38	Nil.
b	1678	916	7.03	"
c	875	418	16.19	"
d	586	307	4.61	"

a. Over 1 cwt. of black mud, containing much organic matter, from Upper Otago Harbor, taken at low tide near Logan's Point.

b. About 2 cwt. of black mud, containing much putrefying organic matter, from the upper harbor, near the outlet of a main sewer.

c. Over 1 cwt. of silt from Pelichet bay, where a small stream of highly ferruginous water runs into the bay from a bed of hematite.

d. About 1 cwt. of silt from the upper end of the harbor, near the outlet of a main sewer from South Dunedin. It contained much decomposing organic matter.

Examination of Wood That Had Been Lying Under Sea-Water for a Long Time.—Twelve samples of wood, which had been buried many years under sea-water and mud in Otago

Analyses of Wood for Gold.

Sample.	Description.	Weight Burned. Pounds.	Results.
a.	Kauri plank, which had lain in Dunedin harbor at least 15 years. Other parts much decomposed.	12	Nil.
b.	Pine plank from Catlin's estuary, east coast of Otago. Part of a ship wrecked 20 years ago.	20	"
c.	Remnant of manuka pile from Otago harbor. Under water 20 years.	12	"
d.	Do., the decayed part.	10	"
e.	Part of a pile from the Vauxhall baths, Anderson's bay, Otago; in the water 30 years.	18	"
f.	Driftwood embedded in the mud at Madagascar beach, west coast of Otago.	16	"

Harbor and elsewhere on the New Zealand coast, were analyzed for gold. In each case from 10 to 30 pounds of the wood was burned, and the ashes were fused with pure litharge.

The results are given on page 617.

Attempts to Precipitate Gold from Sea-Water.

These experiments were confined to such natural reducing-agents as might naturally occur along the coast at the present day, namely, sulphides of iron (chiefly formed by the reduction of the sulphates of sea-water in the presence of iron salts), and carbonaceous matter of various kinds.

To make a suitable filter, an earthenware pipe 4 inches in diameter and 1 foot long was closed at the ends with strong cloth. Next the cloth was placed a loose plug of asbestos, about 2 inches in thickness, wrapped in linen, and the middle part of the pipe was filled with coarsely broken earthenware, with which the reducing agent was mixed. The reducing-agents used were animal charcoal, wood-charcoal, soot, and sulphides of iron, copper and lead—the latter being prepared by precipitating sulphates of iron and copper and the nitrate of lead, in order to make sure that they should contain no gold.

Pelichet Bay is separated from the upper reaches of Otago Harbor by an embankment, in which an opening about 20 feet wide has been left. Through this passage the sea runs with considerable force at most states of the tide. The apparatus above described was fixed beneath the bridge which spans this opening, so that both the ebbing and the flowing current might pass through it. The filter was kept thus immersed for periods varying from one to two months. The reducing-agents were then taken out, roasted and assayed.

The results are given on page 619.

A very large quantity of sea-water must have passed through the apparatus in each of the above cases, but there was no way of estimating it with precision. In view of the negative results of all the experiments, an attempt was made to precipitate gold and silver from a measured quantity of sea-water. Ten thousand grains of artificial sulphides of iron, copper and lead, with animal charcoal and wood-charcoal—all in fine powder—were mixed in a barrel with 60 gallons of sea-water taken from the Pacific Ocean at Tomahawk Head, near Dunedin. The re-

Experiments in Reducing Gold from Sea-Water.

Experiment.	Reducing-Agent.	Remarks.	Gold Found.
a.	Animal charcoal, in fine powder.	The greater part of the animal charcoal had been washed through the filter.* The remainder was assayed.	Nil.
b.	Mixture of pounded charcoal and soot.	The smaller particles washed away. Charcoal remained.	"
c.	Animal charcoal in lumps.		"
d.	Sulphide of iron.	The sulphide was much oxidized; the whole apparatus being coated with ferric oxide.	"
e.	Sulphide of lead in lumps.	Very little altered.	"
f.	Sulphide of copper in lumps.	Carbonate of copper found in the asbestos filter and other parts of the apparatus.	"
g.	Mixture of animal charcoal, soot, sulphide of lead, sulphide of copper and sulphide of iron.	Very little oxidation, even of the artificial sulphide of iron, was noticed.	"

ducing-agents were stirred in the water for half an hour, and the sediment was allowed to settle for some hours. The clear water above was then decanted off, and the barrel was again filled. This operation was repeated fourteen times in the same barrel, or until over 4 tons of sea-water had been treated. The sediment was then collected and roasted at a dull red heat, to incinerate the charcoal and get rid of the sulphur. On assaying the residue, no gold was obtained; but the result was a bead of pure silver weighing 0.0014 grain. I cannot say certainly whether this silver (which contained no gold) came from the sea-water or from the litharge used. I do not think the latter. If it came from the sea-water, however, it is noteworthy that the amount from 4 tons represents only 0.00035

* After the first two attempts the reducing-agents were put in, not as powder, but in lumps from ¼ to ½-inch in diameter, it having been found that the strong current carried the finer stuff through the asbestos filter.

grain per ton, or one five-hundredth part of the quantity found in sea-water by Malaguti and Durocher in 1851.

All my experiments have thus signally failed to show any precipitation of gold (and have practically failed as to silver) from sea-water by natural reagents. So far as they go, they lend no support to the theory that the deposition of gold and silver by such reagents in marine sediments is now going on.

If such deposition had been the rule in former periods, and if this be the origin of the gold in stratified formations, why should only a comparatively small proportion of such formations be traversed by auriferous veins? This point has not escaped the attention of Posepny.* It seems to me that important evidence may be drawn from the examination of stratified rocks known to be consolidated marine sediments, but the lodes in which have not proved auriferous. Table XXIII. gives an examination of nine samples from Skipper's Road, east of Lake Wakatipu, Otago, an area in which the rocks are known to belong to the middle division of the foliated mica-schists (the favorable country-rock for gold in Otago), but in which no auriferous reefs had hitherto been discovered. Only those parts (*e.g.* fillings of veins and seams, etc.) particularly favorable to the deposition of gold were examined, but no gold was found in any of the samples.

A second series of examinations was made on samples from the west shore of Wakatipu lake, which is largely occupied by rocks of the maitai (Carboniferous) and the Te Anau (Upper Devonian) series. These rocks (mostly sandstones and slates) are undoubted marine sediments, but no gold has been found in this area.

If the gold of the lodes in the foliated mica-schists east of this lake was originally deposited from sea-water and has since been collected by lateral segregation, it is difficult to understand why gold should not have been deposited in the marine sediments west of the lake also.

I therefore examined 16 samples of quartz, ferric oxide, etc., from the two series above named, where they are exposed north and south of the Greenstone river. There were 4 samples of fine-grained blue slate; 6 of ferric oxide from joints and fis-

* *Genesis of Ore-Deposits, Trans.*, xxiii., p. 307.

sures; 4 of small quartz-veins, none of which contained sulphides; 2 of slate containing large crystals of pyrite, amounting in weight to 254 grains for the two samples.

No trace of gold or silver was found in any of these samples.

Chapter VIII.—Summary of Results.

[The recapitulation of the various results recorded in the preceding chapters is here omitted, to save space.]

Bearing of these Results on the Origin of Auriferous Lodes.

When I began this work, seven years ago, I was strongly inclined to believe that the lateral-secretion theory afforded the most reasonable explanation of the origin of auriferous deposits in these colonies; but, as the result of each series of examinations appeared, I was forced to the following conclusions:

If any reliance can be placed on the examinations detailed in the foregoing chapters, they seem to indicate that the gold of many lodes of the chief mining districts of New Zealand, Victoria and Queensland is due not to lateral segregation from the adjacent country-rock, but to solutions ascending from some rock deeper than any now exposed at the surface in any part of these colonies.

I am not concerned with the question whether this source is the vague "barysphere," with its somewhat apocryphal contents of heavy metals. I have simply to note that a series of laborious and careful examinations has failed to find it in the rocks of the "lithosphere."

What may be the value of these investigations in the study of the general question of the origin of ore-deposits I leave the reader to judge, being myself content to quote the opinion of Prof. Stelzner, of Freiberg, no mean investigator of that larger question, that

"Each increase of our positive knowledge of the nature and mode of origin of ore deposits, each explanation of any question connected with such deposits or with their associated country-rocks, is a distinct gain, not only to science, but also to mining practice."*

* Concluding sentence of *Die Lateralsecretionstheorie*, by A. W. Stelzner.

TABLE I.—*Analyses of Country-Rock from Deep Levels. New Chum Railway G. M. Co., Bendigo, Victoria.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity above 3.18.	Percentage of Sulphur.	Percentage of Carbon.	Weight of Gold Obtained.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.			Grains.	Grains.
a	At shaft.	277	4.48	240	76	0.06	.....	Nil.	Nil.
b	16	261	4.48	364	42	0.04	0.68	"	"
*c**	37	240	4.48	180	61	0.09	.....	"	"
d	59	218	4.48	424	265	0.57	1.46	.0017	0.85
e	84	193	4.48	356	112	0.24	.....	Nil.	Nil.
*f**	103	174	4.48	180	27	Trace.	0.06	"	"
g	127	150	4.48	456	312	0.63	.....	"	"
h	145	132	4.48	630	418	0.86	1.32	Gold button lost.	.....
h'	145	132	2.24	250	197	0.86	1.32	.0004	0.4
*i**	167	110	4.48	164	26	Trace.	.....	Nil.	Nil.
j	179	98	4.48	212	38	"	.....	"	"
k	191	86	4.48	185	38	"	.....	"	"
l	192	85	4.48	378	216	0.54	Nil.	.0016	0.8
m	192	85	4.48	643	498	1.03	"	.0023	1.15

a. Fine-grained sandstone. No sulphides visible with a hand-lens. Concentrates contained a very small quantity of sulphides.

b. Fine-grained sandy slate. Pyrite in minute cubes, very sparingly distributed throughout the sample. Concentrates contained a small quantity of sulphides.

c. Sandstone somewhat coarse grained. No sulphides visible. Concentrates contained only a trace of sulphides.

d. Fine-grained, comparatively soft, dark-blue slate, from the vicinity of a small slide, showing pyrite and blende sparingly with a hand-lens. Concentrates showed some pyrite and blende.

e. Very hard, fine-grained sandstone. Pyrite in large cubes visible in the sample. Concentrates contained a good percentage of pyrite.

f. Very hard, fine-grained light blue slate; very distinct cleavage; no trace of sulphides. Concentrates contained a trace of sulphides.

g. Rather coarse-grained sandstone, with cubes of pyrite, in one case 15 mm. in diameter. Concentrates nearly all pyrite.

h. Fine-grained dark-blue slate, with a small quantity of pyrite and blende visible with a hand-lens. Concentrates showed a good quantity of pyrite and blende.

h'. Same as above.

i. Very hard, fine-grained sandstone. No sulphides visible. Concentrates contained no sulphides.

j. Same as *i*.

k. Same as *i*.

l. Fine-grained slate from both sides of a "slide," or slight fault, showing pyrite in small quantity. The sample included a little "pug" from the slide. Concentrates nearly all pyrite.

m. Fine-grained slate from both sides of a "slide," or slight fault, showing pyrite in small quantity. The sample included a little "pug" from the slide. Concentrates contained a good percentage of pyrite and blende, with a little galena.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE I.—*Concluded.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity above 3.18.	Percentage of Sulphur.	Percentage of Carbon.	Weight of Gold Obtained.	Yield of Gold per Ton of Country Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.			Grains.	Grains.
n.....	214	63	4.48	387	266	0.58	.....	.0027	1.35
o.....	230	47	4.48	678	429	0.90	0.56	.0017	.85
p.....	250	27	4.48	547	392	1.02	0.68	.0034	1.7
q.....	265	12	4.48	1140	674	1.66	0.89	.0043	2.15
*r**.....	271	6	4.48	146	32	Traces	.....	Nil.	Nil.
s.....	274	3	4.48	1246	827	1.98	2.64	.0065	3.25
t.....	276	1	4.48	1627	1016	2.10	4.50	.0132	6.6
u.....	276	1	4.48	1518	1142	1.87	6.34	.0089	4.45
v *w* *x* ...	277	From the Reef	3.50			3.43	2.16		oz. dwt. gr. 2 16 5 1 0 11 1 15 13

n. Fine-grained sandstone with pyrite in large crystals and a little blende.

o. Dark-blue carbonaceous slate with pyrite, arsenopyrite and blende in fair quantity. Concentrates mostly pyrite and blende.

p. Same as *o*.

q. Same as *o*.

*r**. Fine-grained sandstone. No sulphides visible. Concentrates contained only a trace of sulphides.

s. Hard, dark-blue carbonaceous slate, showing pyrite freely, with a little blende. Concentrates obtained nearly clean sulphides.

t. Dark-colored casing-slate of the lode (almost black in color from carbonaceous matter), showing a large quantity of pyrite, with a little blende and galena. Concentrates nearly clean sulphides.

u. Same as *t*.

v, *w*, *x*. Laminated quartz (laminæ due to carbonaceous matter from the walls), showing pyrite, blende and galena. Not concentrated.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE II.—*Analyses of Country-Rock from a Cross-Cut at the 1180-Foot Level, South St. Mungo G. M. Co., Bendigo, Victoria.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity above 3.18.	Percentage of Sulphur.	Percentage of Carbon.	Weight of Gold Obtained.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.			Grains.	Grains.
a.....	From the Shaft.	128	4.48	316	36	0.015	.....	Nil.	Nil.
b.....	11	117	4.48	468	213	0.52	.....	"	"
c.....	20	108	4.48	314	139	0.31	.....	"	"
d.....	34	94	4.48	275	Nil.	Trace.	.....	Lost.	Lost.
e.....	53	75	4.48	672	246	0.73	0.08	Nil.	Nil.
*f**.....	67	61	4.48	416	28	0.02	Nil.	"	"
g.....	78	50	1000 grs.			0.06	"	"	"
h.....	93	35	4.48	746	394	0.95	Trace.	.0017	0.85
i.....	112	16	4.48	264	98	0.19	1.06	Lost.	Lost.
j.....	124	4	4.48	786	493	1.06	1.38	.0073	3.65
k.....	128	Foot-wall of reef.	4.48	548	391	0.75	4.94	.0124	6.2
l.....	137	In reef 30 feet wide.	1000 grs.			Nil.	Nil.	.0012	18.8
m.....	138	"	1000 grs.			.016	Nil.	.0113	6.7
*n**....	149	Hanging-wall of reef.	4.48	216	84	0.19	Trace	Nil.	Nil.

a. Hard, light-blue sandstone, no sulphides. Contained a quartz vein ¼-inch wide. Concentrates contained only a trace of sulphides.

b. Hard, gray sandstone, showing large cubes of pyrite. Concentrates contained a good percentage of pyrite.

c. Same as above.

d. Fine-grained gray slate, showing no sulphides. Concentrates contained very little pyrite.

e. Fine-grained dark-colored slate, showing small crystals of pyrite. A quartz vein ½-inch wide ran through the slate. Concentrates contained a good percentage of pyrite.

f. Moderately coarse-grained hard sandstone. No trace of sulphides. Concentrates contained very little pyrite.

g. Quartz leader 2 inches wide in sandstone; clear white quartz, showing no sulphides. All assayed.

h. Fine-grained slaty sandstone, with large crystals of pyrite, in some cases 10 mm. in diameter. Concentrates contained a good proportion of pyrite, blende and galena.

i. Dark-colored fine grained carbonaceous slate, showing a very little pyrite.

j. Same as above. Concentrates consisted mostly of sulphides.

k. Carbonaceous slate-casing on foot-wall of reef, with a vein of pyrite about 6 mm. thick running through part of the sample. Concentrates consisted mostly of sulphides.

l. Clean, white, vitreous quartz, with no trace of sulphides. All assayed.

m. Clean, white, vitreous quartz, with small proportion of pyrite and a few crystals of blende and galena. All assayed.

n. Hard, gray slate. No visible sulphides.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE III.—*Analyses of Country-Rock from Cross-Cut* 302 *Feet East at the* 610-*Foot Level, South St. Mungo G. M. Co., Bendigo, Victoria.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity above 3.18.	Percentage of Sulphur.	Percentage of Carbon.	Weight of Gold Obtained.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Pounds.	Grains.	Grains.			Grains.	Grains.
a	From Shaft.	42	4.48	212	48	Trace.	.....	Nil.	Nil.
b	48	90	4.48	316	66	"	.....	"	"
c	75	117	1000 grs.	.....	.....	"	.....	"	"
d	119	161	4.48	716	342	0.47	.....	.0003	0.15
*e**	176	218	4.48	205	116	0.18	.....	Nil.	Nil.
f	246	288	4.48	182	46	Trace.	.....	"	"
*g**	302	344	4.48	210	94	0.24	.....	"	"

a. Hard, fine-grained sandstone. No visible sulphides. Concentrates contained very little pyrite.

b. Same as *a*.

c. White leader of quartz, 3 inches wide, in hard sandstone country, showing no sulphides. All assayed.

d. Hard, fine-grained sandstone, with large crystals of pyrite, 8 mm. in diameter. Concentrates contained good percentage of sulphides.

e. Hard, dark-gray slate. No visible sulphides. Concentrates contained very little pyrite.

f Hard sandstone with small, clear, white vein of quartz about ¼-inch wide. No sulphides visible. Concentrates contained very little pyrite.

g. Same as *f*.

In the case of samples marked * the whole of the sample left after concentration was assayed after concentrates had been examined.

TABLE IV.—*Examination of Samples from the Prince Regent G. M. Co., Ballarat East.*

A. *Country-Rock.*

Sample.	Distance from Indicator.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Yield of Gold.	Yield of Gold per Ton.
	Feet.		Grains.	Grains.	Grains.	Grains.
a	20 East.	4.48	165 No sulphides.	28	Nil.	Nil.
b	14 "	4.48	217	35	"	"
c	10 "	4.48	194	83	"	"
d	4 West.	4.48	370 With a little pyrite.	216	.007	0.35
e	16 "	4.48	116 No sulphides.	24	Nil.	Nil.
f	25 "	4.48	136 "	49	"	"
	Distance from Mundic Slate.					
g	16 West.	4.48	212 "	53	"	"
h	33 "	4.48	327 With a good percentage of pyrite.	205	"	"
i	68 "	4.48	173 No sulphides.	32	"	"
j	120 "	4.48	417 With a good percentage of pyrite.	236	"	"

a. Hard, solid slate, showing no sulphides.

b. Slate, showing no sulphides.

c. Sandstone, showing a little pyrite.

d. Solid, fine-grained slate near a quartz-vein crossing the Indicator; showed a very little sulphide.

e. Solid slate, showing no pyrite.

f. Solid sandstone from a stratum 6 inches wide; showed no pyrite.

g. Solid slate from the middle of a 6-foot stratum of slate.

h. Solid sandstone showing a little pyrite in large cubes.

i. Hard, solid slate from band 9 inches wide between two sandstone beds.

j. Solid sandstone, showing a little pyrite in large cubes.

TABLE IV.—*Concluded.*

B. Pug, Etc., From Fault-Lines.

Sample.	Depth Below Surface.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Yield of Gold.	Yield of Gold per Ton.
	Feet.	Lbs.	Grains.	Grains.	Grains.	Grs.
1.....	630	2.24	673 With a fair percentage of pyrites.	218	0.0158	15.8
2.....	"	1.12	416 Nearly all pyrite.	309	0.013	26
3.....	"	2.24	433 With a good proportion of sulphides.	289	0.0062	6.2
4.....	"	2.24	248 Showed a little pyrite.	117	Nil.	Nil.
5.....	"	1.12	490 Nearly clean sulphides.	313	0.0075	15
6.....	From 770-foot Level.	2.24	676 do. do. do.	448	0.0113	11.3
7.....	"	2.24	379 do. do. do.	216	0.010	10
8.....	"	1.12	256 Nearly all pyrite.	190	0.0009	0.45
9.....	"	2.24	134 Nearly clean pyrite.	98	0.0176	17.6

1. From a slide or "back" 23 feet W. of the Indicator: "pug" or clay, colored dark by a large quantity of carbonaceous matter.

2. The same, from the same slide higher up: stiff bluish clay, with large crystals of pyrite and a little galena.

3. Broken slate, "pug" and quartz from a slide or "back," 10 feet E. of the mundic slate; sample showed a little blende and a fair quantity of pyrite.

4. Broken country-slate, sandstone and quartz from an extensive fault 90 feet W. of the mundic slate.

5. "Pug" from a slight fault, against which the quartz-veins terminated, 15 feet W. of the mundic slate. Showed a large quantity of pyrite, with scattered crystals of galena and blende.

6. "Pug" and broken slate and sandstone, with quartz-veins, from an extensive fault which threw the Indicator about 40 feet to the W. Sample showed a large proportion of carbonaceous matter, with a little pyrite and galena.

7. From the same sample, higher up in the drive.

8. From a slide to the W. of a vein crossing the Indicator. Sample taken 20 feet W. of the Indicator: quartz-vein, with pyrite and much dark carbonaceous matter.

9. From a slide 16 feet E. of the Indicator, consisting of blue "pug," showing a little pyrite and blende.

TABLE V.—*Analyses of Country-Rock from Deep Levels, Northern Star G. M. Co., Ballarat West.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Pounds.	Grains.	Grains.		Grains.	Grains.
a	0	100	4.48	253	48	0.002	Nil.	Nil.
b	37	63	4.48	416	50	0.017	.0002	0.1
c	81	19	4.48	378	89	0.035	.0004	.2
*d**	92	8	4.48	210	23	Nil.	Nil.	Nil.
e	99	1	4.48	1217	876	1.46	.0071	3.55
f	100	"Guiding Star" lode.	1000 grs.	.....	.....	2.8	.....	314.
g	104	0	4.48	586	347	.37	.0036	1.8
*h**	131	27	4.48	216	49	Trace.	Nil.	Nil.
i	174	70	4.48	315	42	"	"	"
j	216	112	4.48	842	194	.37	"	"
k	216	112	1000 grs	.....	.....	Trace.	"	"
l	278	174	4.48	624	416	.62	"	"
*m**	336	232	4.48	317	46	.02	"	"
n	399	295	4.48	426	240	.41	.0013	0.65

a. Hard, fine-grained sandstone with vein of white quartz ½-inch wide. No sulphides visible.

b. The same, but with a small quantity of pyrites in large cubes. Concentrates contained a good percentage of pyrite.

c. Fine-grained blue slate, showing a little pyrite. Concentrates contained a little galena.

d. Hard, grayish-blue sandstone. No pyrites visible.

e. Carbonaceous slate with small quartz-veins, forming the casing. Concentrates nearly all sulphides.

f. White, splintery quartz from the "Guiding Star" lode, with laminæ of carbonaceous shale, showing a little pyrite and arsenopyrite. All analyzed.

g. Fine-grained black slate from the wall of the "Guiding Star" lode. Showed a very little pyrite and arsenopyrite. Concentrates mostly sulphides.

h. Hard sandstone with no sulphides.

i. The same.

j. Slate from beside a small quartz-vein 2 inches wide. Showed a little pyrite. Concentrates contained fair proportion of pyrite.

k. White quartz from leader 2 inches wide. No sulphides visible. All analyzed.

l. Sandstone with large cubes of pyrite from hanging-wall of a vein of quartz about 6 inches wide. Concentrates contained a large percentage of sulphides.

m. Sandstone, hard, moderately coarse-grained. No sulphides.

n. Slate, with pyrite and arsenopyrite visible in small quantities, taken from the wall of a slide which was filled with recently-formed limonite. Concentrates showed a good percentage of sulphides.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE V.—*Continued.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
o	440	336	4.48	.....	.....	.....	.....	.....
p	499	395	4.48	435	193	0.32	.0027	1.35
q	499	395	1000 grs.			0.68	.0016	25.
r	550	446	4.48	1216	800	1.06	Nil.	Nil.
s	617	513	4.48	316	74	Trace.	"	"
t	678	574	4.48	743	495	0.79	.0071	3.55
u	741	637	.....		.....	.....	.....	.....
v	804	700	4.48	216	39	Nil.	Lost.	Lost.
*w**	859	755	4.48	349	72	"	Nil.	Nil.
x	908	813	.....		.....	.....	.....	.....
y	971	867	4.48	614	394	0.73	"	"
z	1017	903	4.48	1026	859	1.37	.0015	0.75
A	1051	947	4.48	694	512	0.78	Nil.	Nil.
B	1060	956	4.48	1846	1327	2.35	.0008	0.4

o. Solid, hard sandstone. No pyrite visible. Concentrates not analyzed.

p. Slate from below a small flat quartz-vein 2 inches wide. Showed a little sulphide. Concentrates contained a good percentage of pyrite.

q. Quartz from a small flat vein, showing a little pyrite.

r. Slaty sandstone, showing large crystals of pyrite. Concentrates nearly pure pyrite.

s. Slate, with no sign of pyrite, from near a small quartz-vein, varying from 8 inches to 1 inch in width.

t. Slate very close to a fault or "slide," showing pyrite and blende. Concentrates contained a large percentage of sulphides.

u. Hard, fine-grained sandstone, showing no sulphides. Concentrates not analyzed.

v. Same as above.

w. Fine-grained slate, showing no trace of sulphides.

x. Hard, fine sandstone. No pyrite. Concentrates not analyzed.

y. Sandstone from the foot-wall of a lode 1 foot wide, running parallel with the "Guiding Star." Showed pyrite in large crystals. Concentrates contained good proportion of pyrite.

z. Slaty sandstone, with crystals of pyrite and a small proportion of arsenopyrite. Concentrates nearly all sulphides.

A. Sandstone with large crystals of pyrite. Concentrates nearly all pyrite.

B. Slate from the wall of an eruptive dike of felsite porphyry. Large crystals of pyrite were abundant in the sample, and blende and galena sparingly distributed.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE V.—*Concluded.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
Dike 1	1068	964	4.48	2143	1827	3.69	.0134	6.7
" 2	1073	969	4.48	946	743	0.96	.0076	3.8
" 3	1082	978	4.48	176	34	Trace.	Nil.	Nil.
" 4	1095	991	4.48	1875	1642	3.24	.004	2.0
C	1097	993	4.48	1163	901	1.73	.006	3.0
D	1150	1046	4.48	314	86	0.18	Lost.	Lost.
E	1207	1103	4.48	946	713	1.02	.0075	3.75
F	1221	1117	4.48	673	442	0.72	.014	7.0
G	1242	1138	4.48	476	319	0.54	.0035	1.75

1. Sample taken from 4 feet in the dike. Contained large quantity of pyrite in large crystals, with inclusions of slate and sandstone. Concentrates nearly all sulphides.

2. The same, with much less pyrite. Concentrates nearly all sulphides.

3. The same. No sulphides visible, and no inclusions of slate and sandstone.

4. Dike 1 foot from farther wall, included many fragments of slate and sandstone from the country-rock. Contained pyrite in large quantities, with a little arsenopyrite and blende.

C. Sandstone from the farther wall of dike, containing large crystals of pyrite. Concentrates mostly pyrite.

D. Sandstone from the vicinity of a "slide." No pyrite visible.

E. Slate from the wall of a small quartz-vein, 6 inches wide. Showed a fair quantity of pyrite with a little galena. Concentrates nearly all sulphides.

F. Sandstone, with large crystals of pyrite. Concentrates mostly pyrite.

G. Slate, with a little pyrite and blende, from the farthest point of the workings.

TABLE VI.—*Analyses of Country-Rock from Deep Levels, United Albion G. M. Co., Steiglitz, near Ballarat, Victoria.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
a	0	69	4.48	276	48	Nil.	Nil.	Nil.
b	29	40	1000 grs.			"	"	"
c	51	18	4.48	647	321	0.73	.0053	2.65
d	68	1	4.48	1035	914	1.037	.0119	5.95
e	76	1	4.48	719	641	.846	Lost.	Lost.
f	93	17	4.48	346	285	.436	.0018	0.9
g	120	44	4.48	517	312	.56	.0023	1.15
h	140	64	4.48	193	35	Nil.	Nil.	Nil.
*i**	169	93	4.48	276	43	"	"	"
j	203	127	4.48	318	58	"	"	"
k	220	144	4.48	725	416	.71	.0029	1.45
*l**	268	192	4.48	186	75	Nil.	Nil.	Nil.
m	307	231	4.48	247	36	"	"	"

a. Slate, no visible sulphides.

b. White, barren-looking quartz from a large "blow" of quartz from 8 to 10 feet through, showing no sulphides. All assayed.

c. Sandstone, showing a small percentage of sulphides. Concentrates contained good proportion of sulphides.

d. Dark-colored slate from the foot-wall of an auriferous lode, showing a good deal of pyrite and arsenopyrite. Concentrates nearly pure sulphides.

e. Slate from the hanging-wall of the same lode. Concentrates nearly all sulphides.

f. Fine-grained sandstone, with large crystals of pyrite. Concentrates nearly all sulphides.

g. Dark-colored slate with a little pyrite. Concentrates contained good percentage of sulphides.

h. Hard, fine-grained, gray slate, with small white quartz-veins. No sulphides visible.

i. Hard, fine-grained sandstone. No pyrite visible. Concentrates contained no visible sulphides.

j. Hard slate. No sulphides visible in sample or concentrates.

k. Dark slate from the anticlinal axis Showed several small veins of quartz, with a little pyrite. Concentrates contained a good percentage of pyrite.

l. Sandstone, with small quartz-veins. No sulphides visible, and none in concentrates.

m. Hard sandstone. No sulphides visible in sample or concentrates.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE VI A.—*Analyses of Samples from Deep Levels, Tam o'Shanter G. M. Co., Steiglitz, near Ballarat, Victoria.*

Sample.	Distance West of Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.	Feet.	Pounds.	Grains.	Grains.		Grains.	Grains.
a*	185	100	4.48	375	46	Trace.	Nil.	Nil.
b	160	75	4.48	436	36	"	"	"
c	131	46	4.48	724	546	0.693	"	"
d	95	10	4.48	376	48	Trace.	"	"
e	85	0	4.48	276	38	"	"	"
f	83	0	1000 grs.	.....	.....	0.469	"	75.6
g	80	0	4.48	943	816	1.23	0.0097	4.85
h	59	21	4.48	612	472	0.76	0.0013	0.65
i	31	49	4.48	573	389	0.48	0.0024	1.2
j	2	78	4.48	674	513	0.63	0.0053	2.65
	East of Shaft.							
k	6	57	4.48	417	346	0.47	Lost.	Lost.
l	29	34	4.48	357	291	0.46	0.0016	0.8
m*	48	15	4.48	213	24	Nil.	Nil.	Nil.
n	61	2	4.48	817	494	0.87	0.011	5.5
o	80	17	4.48	246	42	Trace.	Nil.	Nil.
p	91	28	4.48	378	241	0.35	0.0021	1.05

a Fine grained sandstone. No sulphides.

b. Fine grained slate showing no sulphides.

c Fine-grained dark slate, showing small quantity of pyrite. Concentrates contained a good percentage of pyrite.

d. Slaty sandstone, showing no sulphides.

e. Casing-slate of a lode filling a fault-fissure which has evidently faulted the country considerably. Sample showed no sulphides.

f. Part of lode consisting of white quartz with small proportion of sulphides. All assayed.

g. Dark, fine-grained slate forming the casing of the lode on the E. side. Pyrite visible in good proportion. Concentrates nearly pure sulphides.

h. Slate with fine threads of quartz crossing the bedding-planes. A little pyrite and blende in the quartz. Concentrates nearly pure sulphides.

i. Similar slate, from the synclinal axis. Concentrates contained good percentage of sulphides.

j. Slate showing good proportion of sulphides. Concentrates nearly all pyrite.

k. Sandstone with large crystals of pyrite. Concentrates nearly all pyrite.

l. Dark-blue sandy-slate, showing no sulphides. Concentrates contained a fair quantity of pyrite.

m. Hard, gray sandstone. Showed no sulphide.

n. Sandstone with a small quantity of pyrite. Concentrates contained a fair proportion of pyrite.

o. Hard, gray sandstone. No trace of sulphides.

p. Slate, with a small proportion of sulphides. Concentrates contained a good percentage of sulphides.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE VII A.—*Long Tunnel G. M. Co., Gippsland, Victoria (Upper Silurian). Analyses of Samples from Dike not Associated with an Auriferous Reef.*

Sample.	Weight Examined.	Weight of Concentrates.		Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Lbs.	Grains.			Grains.	Grains.
a	7	1650	Showing no sulphides.	Nil.	Nil.	Nil.
b	7	650	Showing no sulphides.	"	"	"
c	7	860	Showing a small quantity of sulphides.	0.73	0.0013	0.416
d	7	647	Showing no sulphides.	Nil.	Nil.	Nil.
e	7	1612	Showing a little pyrite.	0.86	0.0016	0.512

a. From the N.W. level 900 feet below the tunnel and about 1600 feet below the surface, from a dike about 3 feet wide, not associated with an auriferous reef. The quartz in the vicinity was quite white in color and non-gold-bearing.

b. The same, but 10 feet away from the last sample.

c. The same, but contained arsenopyrite in needles, sparingly distributed through the sample, with a little pyrite.

d. 36 feet from last sample from the hanging-wall of a quartz-reef, which contained no sulphides and was not auriferous.

e. Formed the middle of a lode between two veins of clear white quartz; non-auriferous; contained a very little arsenopyrite in needles.

TABLE VII B.—*Analyses of Samples from Dike Associated with Rich Auriferous Reef.*

Sample.	Weight Examined.	Weight of Concentrates.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Lbs.	Grains.		Grains.	Grains.
a	4.48	2107 With a good proportion of sulphides.	1.86	0.049	21.5
b	4.48	617 Mostly arsenopyrite.	0.95	0.062	31.
c	4.48	1094 With a good proportion of sulphides.	1.36	0.043	21.5

a. From the 1422-foot level, about 2100 feet below the surface, forming the middle of a richly auriferous lode. Sample heavily impregnated with pyrite, and arsenopyrite in needles. (The quartz associated with it yielded 2 oz. 3 dwts. per ton)

b. From the 1322-foot level, about 2000 feet below the surface, forming the footwall of a rich lode about 4 feet wide. The dike showed very slight traces of sulphides.

c. From the 1422-foot level, about 2100 feet below surface, forming the hangingwall of a rich lode, the lode being 18 inches wide, and the dike about 2 feet 4 inches wide, the dike-rock richly impregnated with pyrite and arsenopyrite.

TABLE VIII.—*Analyses of Country-Rock of Walhalla, from Cross-Cut at 900-Foot Level (about 1600 Feet Below Surface).*

Sample.	Distance from Lode Fissure.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity above 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
a.....	278 from dike.	4.48	564 With very small percentage of sulphides.	56	Trace.	Nil.	Nil.
b.....	231	4.48	326	25	"	"	"
c..	178	4.48	637	218	0.373	"	"
d.....	178	2.24	416 Nearly all sulphides.	374	0.56	"	"
e.....	125	4.48	274 No sulphides.	39	Nil.	"	"
f.....	97	4.48	624 Mostly sulphides.	490	0.61	0.0008	0.4
*g**.....	74	4.48	447 With a good percentage of pyrite.	276	0.37	Nil.	Nil.
h.....	32	4.48	394 With a fair per centage of sulphides.	134	0.17	"	"
i.....	14	4.48	439 With a good proportion of sulphides.	202	0.34	"	"
j.....	1	4.48	364	42	Trace.	"	"

a. At shaft. Very hard sandstone, showing no sulphides.

b. Hard, fine-grained slate, showing no sulphides.

c. Hard sandstone with quartz-veins, from the foot-wall of a small slide or fault. A very little arsenopyrite visible in sample.

d. Broken slate and sandstone from the same slide, with small "stringers" of quartz; showed a little pyrite and arsenopyrite.

e. Very hard, dense, slaty sandstone from a bar about 6 inches wide, interbedded with the country.

f. Fine-grained slate, with quartz-seams crossing the bedding-planes, from a disturbed part of the country. A little pyrite and arsenopyrite visible.

g. Sandstone, somewhat coarse-grained, with a few large crystals of pyrite.

h. Slaty sandstone, fine-grained, with small needles of arsenopyrite visible sparingly in the sample.

i. Dark, fine-grained slate, with arsenopyrite in needles, sparingly distributed throughout, and a little pyrite.

j. Hard sandstone, within 1 foot of dike 5 feet wide. The dike in this case fills the whole fissure, and contains neither quartz nor gold.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE IX.—*Analyses of Samples from the* 1422-*Foot Level, from a Cross-Cut Driven* 165 *Feet East to the Boundary of the Long Tunnel Extended Company.*

Sample.	Distance from Dike.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.	Lbs.		Grains.		Grains.	Grains.
a	165	4.48	375 No sulphides.	39	Trace.	Nil.	Nil.
b	139	2.24	987 With a fair quantity of sulphides.	642	1.03	"	"
*c**	102	4.48	247 No sulphides.	32	Trace.	"	"
d	75	4.48	312 Very little pyrite.	83	"	"	"
e	43	4.48	417 With a good percentage of pyrite.	180	0.22	0.0014	0.7
f	25	4.48	232 No sulphides.	37	0.07	Nil.	Nil.
g	13	4.48	643 With a good percentage of sulphides.	519	0.86	0.0062	3.1
h	1	4.48	408 Mostly sulphides.	315	0.63	0.0081	4.5

a Slate, with quartz-veins. No sulphides visible.

b. Crushed slate from a small slide or fault. Sample contained a little quartz, much oxidized, but showing pyrite sparingly.

c. Hard, rather coarse-grained sandstone. Showed no pyrite.

d. Very dense gray sandstone in a band 4 inches wide between two beds of slate. Showed no sulphides.

e. Hard, gray slate, with a few small crystals of pyrite.

f. Dark slate from wall of small slide filled with quartz. The sample included a little quartz. No sulphides visible.

g. Hard, gray sandstone, with crystals of pyrite and arsenopyrite, and arsenopyrite in needles.

h. Slate forming foot-wall; very fine-grained. Showed a little pyrite and arsenopyrite.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE X.—*Analyses of Country-Rock from Deep Levels, No. 5, North Phœnix G. M. Co., Gympie, Queensland.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
a	75	126	4.48	674	193	Trace.	Nil.	Nil.
b	150	51	4.48	426	317	0.56	0.0027	1.35
c	200	0	4.48	1273	1096	1.87	0.0146	7.3
d	203	0	4.48	647	519	0.94	0.0217	10.85
e	314	113	4.48	213	79	0.06	Nil.	Nil.
f	420	194	4.48	476	312	0.43	"	"
g	474	150	4.48	519	426	0.73	"	"
h	562	52	4.48	657	317	0.64	Lost.	Lost.
i	588	26	4.48	1016	842	1.37	0.0273	13.65
j	613	0	4.48	1273	990	2.17	0.0196	9.8
k	613	0	4.48	513	391	0.74	0.0106	5.3

a. Purple conglomerate. Concentrates contained a small percentage of sulphides.

b. Purple conglomerate, from the wall of a fault hading east at an angle of 43°. Sulphides were easily seen in parts of sample. Concentrates nearly all pyrite.

c. Black carbonaceous shale forming the casing on the hanging-wall of an auriferous reef 2 feet wide; pyrite in large quantity visible in sample. Concentrates mostly pyrite, containing traces of galena and blende.

d. The same, from foot-wall of reef; sulphides present in less quantity than in the last.

e. Black carbonaceous shale. No sulphides visible. Concentrates contained only a trace of sulphides.

f. The same, showing a little pyrite. Concentrates contained a good percentage of pyrite.

g. Same as above.

h. The same. No sulphides visible.

i. The same, showing a large quantity of pyrite, with little blende.

j. Dark carbonaceous shale forming the hanging-wall of an auriferous reef. The sample showed a large quantity of pyrite. Concentrates nearly pure pyrite.

k. The same. Much less pyrite in sample. Concentrates nearly pure pyrite.

TABLE XI.—*Analyses of Country-Rock from Deep Levels, Brilliant and St. George G. M. Co., Charters Towers, Queensland.*

Sample.	Distance from Shaft.	Distance from Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.		Grains.	Grains.		Grains.	Grains.
a.....	60	2	4.48	1094	816	1.132	0.0172	8.6
b.....	60	2	4.48	476	126	0.069	0.0013	0.65
*c**.....	38	24	4.48	349	113	Trace.	Nil.	Nil.
d.....	38	24	4.48	276	90	Nil.	"	"
*e**.....	15	47	4.48	394	146	"	"	"
f.....	15	47	4.48	306	125	"	"	"
*g**....	0	62	4.48	417	109	"	"	"
h.....	0	62	4.48	233	87	"	"	"
								Average: Oz. Dwt. Grs.
i.....	62	From reef.	1000 grs.					2 6 9
j.....	62	"	"					
k.....	62	"	"					

a. Tonalite, with a little black mica. A vein of pyrite containing scattered crystals of blende and galena ran through the sample. Concentrates nearly all clean pyrite.

b. The same. Showed no pyrite. Concentrates contained very little pyrite.

c. The same. No pyrite visible.

d. The same.

e. The same.

f. The same.

g. The same.

h. The same.

i. Quartz, with large percentage of pyrite, galena and blende. All assayed.

j. All assayed.

k. All assayed.

In the case of samp'es marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE XII.—*Analyses of Quartz-Folia from the Mica-Schists of the Tipperary and Premier Mines, Macetown, Otago.*

Sample.	Distance from Gold-Bearing Reef.	Weight Analyzed.	Weight of Concentrates.	Weight of Concentrates Having Specific Gravity over 3.18.	Sulphur as Insoluble Sulphides.	Yield of Gold.	Yield per Ton of Rock.
	From Dublin United reef, Tipperary Co.						
	Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
A...	18	2.24	473	135 Cont'd very little pyrite.	0.096	0.0003	0.3
B...	26	2.24	580	419 Nearly clean pyrite.	0.63	0.0024	2.4
C...	113	2.24	312	110	Trace.	Nil.	Nil.
D...	207	2.24	409	212	0.12	"	"
E...	426	2.24	248	97	Trace.	"	"
	From Killarney reef, Tipperary Co.						
F...	360	2.24	618	377	0.56	0.0004	0.4
G...	217	2.24	249 Very little pyrite.	103	Trace.	Nil.	Nil.
H...	88	2.24	217 No pyrite visible.	92	Nil.	"	"
I...	48	2.24	283 About half pyrite.	178	0.37	0.0013	1.3
K...	5	2.24	410 Mostly pyrite.	290	0.75	0.0032	3.2
L.*	48	2.24	317 No sulphides visible.	89	Nil.	Lost.	Lost.
M...	87	2.24	490 About half pyrite.	276	0.49	0.0013	1.3
	From Premier lode, Premier Co.						
N...	16	2.24	307 N'ly clean pyrite.	283	0.68	0.0069	6.9
O...	32	2.24	294 N'ly clean pyrite.	187	0.32	Lost.	Lost.
P*..	9	2.24	287 Very little pyrite.	43	Nil.	Nil.	Nil.

A. From quartz-folia about 4 inches wide, dark-colored, with bands of mica. No sulphides visible with hand-lens. B. From about 3 inches wide. Pyrite and arsenopyrite visible sparingly with hand-lens, along foliation-planes. C. From about 6 inches wide. No sulphides visible. D. From about 4 inches wide. No sulphides visible. E. From about 3 inches wide. No sulphides visible. F. From about 4 inches wide. Pyrite visible in small quantity. G. From about 3 inches wide. Much mica in foliation-planes, but no pyrite visible. H. From about 4 inches wide. Clean white quartz. No sulphides. I. From about 3 inches wide. Showed pyrite in veins, with mica. K. From 2 inches wide. Dark bands running parallel to foliation-planes, and showing good proportion of pyrite and arsenopyrite. L. From about 6 inches wide. White quartz. No sulphides visible. M. From about 3 inches wide, mica in foliation-planes, with pyrite in small quantity. N. From about 2 inches wide, showing pyrite in veins. O. From about 3 inches wide, showing pyrite in small quantity. P. From about 2 inches wide. Clean white quartz, showing no sulphides. In the case of samples marked * the whole of the sample after concentration was assayed after the concentrates had been examined.

TABLE XIII.—*Analyses of Samples of Country-Rock from the Deep Circulation of the Tipperary G. M. Co., Macetown (Low Level Tunnel).*

Sample.	Distance from Gold-Bearing Reef.	Weight Analyzed.	Weight of Concentrates.	Weight of Concentrates Having Specific Gravity over 3.18.	Sulphur as Insoluble Sulphides.	Yield of Gold.	Yield of Gold per Ton of Rock.
	From Dublin United Reef. Feet.	Lbs.	Grains.	Grains.	Grains.	Grains.	Grains.
a	16	4.48	317 With little pyrite.	109	0.16	0.0003	0.15
b	37	4.48	349 No pyrite	37	Trace.	Nil.	Nil.
c	62	4.48	427 With a fair quantity of pyrite.	218	0.39	0.0017	0.85
d	90	4.48	248 No sulphides.	20	Nil.	Nil.	Nil.
e	130	4.48	173 " "	43	"	"	"
f	158	4.48	386 " "	71	Trace.	Lost.	Lost.
g	198	4.48	318 " "	59	Nil.	Nil.	Nil.
h	245	4.48	209 " "	47	"	"	"
i	273	4.48	373 A fair percentage of pyrite.	219	0.36	"	"
j	326	4.48	597 A good proportion of sulphides.	373	0.58	"	"
k	405	4.48	630 Pyrite in fair proportion.	296	0.30	"	"
l	438	4.48	375 Nearly all sulphides.	280	0.41	"	"
	From Killarney Reef.						
m.....	400	4.48	209 Very little pyrite.	33	Trace.	Lost.	Lost.
n	317	4.48	173 " " "	49	Nil.	Nil.	Nil.
o	250	4.48	419 A good quantity of sulphides.	317	0.59	0.0007	0.35

a Mica-schist. No sulphides visible.
b. Same as above.
c. The same. Pyrite visible in small quantity in quartz-folia.
d. The same. No pyrite visible.
e. Same as above.
f. Same as above.
g. The same, from vicinity of slight fault. The sample included a small quantity of fault rock.
h. The same. No sulphides visible.
i. The same, from a slight fault. Sample much broken and contorted, with quartz veins in all directions, showing a little pyrite.
j. Solid mica-schist, with a little pyrite and arsenopyrite in foliation-planes and joints.
k. Mica-schist, with a little pyrite and arsenopyrite in foliation-planes and joints.
l. The same, showing pyrite and arsenopyrite pretty freely in quartz-folia.
m. 940 feet from entrance of tunnel. Mica-schist. No sulphides visible.
n. Mica-schist. No sulphides visible.
o. Broken and contorted mica-schist from a fault-line, containing small veins of quartz, which showed pyrite and arsenopyrite in fair quantity.

TABLE XIII.—*Concluded.*

Sample.	Distance from Gold-Bearing Reef.	Weight Analyzed.	Weight of Concentrates.	Weight of Concentrates Having Specific Gravity over 3.18.	Sulphur as Insoluble Sulphides.	Yield of Gold.	Yield of Gold per Ton of Rock.
	From Killarney Reef. Feet.	Lbs.	Grains.	Grains.		Grains.	Grains.
p....	217	4.48	265 No pyrite.	59	Nil.	Nil.	Nil.
q....	175	4.48	290 With a little pyrite.	111	0.19	0.0013	0.65
r....	143	4.48	173 No sulph'ds.	31	Lost.	Lost.	Lost.
s....	110	4.48	149 "	38	Nil.	Nil.	Nil.
t....	74	4.48	304 With good proportion of sulph'ds.	209	0.35	0.0004	0.2
u....	39	4.48	237	53	Trace.	Nil.	Nil.
v....	18	4.48	137 With very sm'll qn'ty of sulph'ds	48	0.09	0.0062	3.1
w....	Foot-wall of auriferous lode, Killarney Reef.	4.48	643 Good prpt'n of sulph'ds	237	0.39	0.0029	1.45
x....	Hanging-wall of auriferous lode, Killarney Reef.	4.48	509 Good prpt'n of sulphd's.	340	0.58	0.0004	0.2
	From Killarney Reef.						
y....	21	4.48	168 Very little pyrite.	74	0.03	Nil.	Nil.
z....	49	4.48	340 With a good percentage of sulphd's.	194	0.37	0.0009	0.45
a'...	83	4.48	293 No sulph'ds.	47	Nil.	Nil.	Nil.
b'...	176	4.48	178 Small qant'y of sulphd s.	73	0.011	0.0002	0.1
c'...	240	4.48	216 No sulphd's.	29	Nil.	Nil.	Nil.
d'...	277	4.48	317 Very small quantity of sulphides.	83	Trace.	0.0005	0.25
e'...	300	4.48	149 No sulph'ds.	24	Nil.	Nil.	Nil.
f'...	312	4.48	145 "	43	"	Lost.	Lost.
g'...	326 From farther end of tunnel, 1530 feet from entrance.	4.48	317 "	98	"	Nil.	Nil.

p. 740 feet from tunnel-entrance. Solid mica-schist. No sulphides visible. *q*. Solid mica-schist, with small quantity of pyrite, visible especially in quartz-folia. *r*. Solid mica-schist. No sulphides visible. *s*. Solid mica-schist. No sulphides visible. *t*. Solid mica-schist, with pyrite in foliation-planes with mica. *u*. Solid mica-schist. No sulphides visible. *v*. Solid mica schist, showing no pyrite. *w*. Mica-schist, much broken and with numerous veins of quartz and calcite, the quartz showing arsenopyrite and pyrite freely. *x*. Mica-schist, much broken and with numerous veins of quartz and calcite, the quartz showing arsenopyrite and pyrite freely. *y*. Solid mica-schist. Very little pyrite. *z*. Solid mica-schist, showing sulphides rather freely in foliation planes, with mica. *a'*. Solid mica-schist. No sulphides visible. *b'*. Solid mica-schist, with small quantity of sulphides in quartz-folia. *c'*. Solid mica-schist. No sulphides visible. *d'*. Mica-schist much broken, with veins of quartz running through sample. No sulphides visible. *e'*. Solid mica-schist. No sulphides visible. *f'*. Solid mica-schist. *g'*. Solid mica-schist.

TABLE XIV.—*Examination of Samples from the Deep Circulation of Macetown Reefs, the Samples Being Taken near the Reef.*

Sample.	Distance from Auriferous Lode.	Locality.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.		Lbs.	Grains.	Grains.		Grains.	Grains.
a	2	Low-level tunnel, Premier Mine, Macetown, Otago.	4.48	716	309	0.65	0.0006	0.3
b	1	"	4.48	642	184	0.37	0.0003	0.15
c	1	"	4.48	317	54	Nil.	Nil.	Nil.
d	12	"	4.48	512	234	0.38	"	"
e	36	"	4.48	440	86	0.06	"	"
f	65	"	4.48	763	219	0.39	0.0002	0.1
g	12	High-level adit, Premier Mine.	4.48	590	347	0.58	0.0017	0.85
h	32	"	4.48	463	185	0.34	0.0008	0.4
i	40	"	4.48	576	429	0.86	0.0075	3.75

a. Hanging-wall country, solid mica-schist with a little pyrite and arsenopyrite in veins. Concentrates contained a good percentage of pyrite.

b. Hanging-wall country of a part of the lode where the lode-fissure was filled with "mullock." Sample was much broken and showed a little pyrite.

c. Foot-wall country of a part of the lode where the lode-fissure was filled with "mullock." Solid mica schist. No sulphides visible.

d. From a cross cut in hanging-wall country; the country much broken. Sample showed no sulphides. Concentrates contained a little sulphide.

e. In the same cross-cut, 24 feet further in. Solid mica schist. The specimen contained veins of quartz, with no visible pyrite. Concentrates contained a little pyrite.

f. Farther in, in the same cross-cut. Sample showed a little pyrite in the interlaminations, associated with mica.

g. In foot-wall of broken country. The sample contained much quartz, with a little pyrite and arsenopyrite. Concentrates contained a good percentage of sulphides.

h. Twenty feet farther in, in the same cross-cut. Broken country, showing a little pyrite and arsenopyrite. Concentrates contained a good percentage of sulphides.

i. Eight feet farther in, in the same cross-cut. Showed a considerable quantity of arsenopyrite and pyrite.

TABLE XIV.—*Continued.*

Sample.	Distance from Auriferous Lode.	Locality.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.		Lbs.	Grains.	Grains.		Grains	Grains.
j	5	High-level adit, Premier Mine.	4.48	609	386	0.67	Nil.	Nil.
k	19	"	4.48	375	193	0.29	0.0007	0.35
l	35	"	4.48	246	31	Nil.	Nil.	Nil.
m	68	"	4.48	309	54	"	"	"
n	89	"	4.48	376	300	0.52	0.0095	4.75
o	98	"	4.48	345	272	0.45	0.0111	5.55

j. Over 5 feet in Moyle's cross-cut. Broken country from hanging-wall side. The reef-fissure at this point contained no quartz. Sample contained a fair quantity of pyrite.

k. From the same cross-cut, 14 feet farther in.

l. From the same cross-cut. Solid country, opposite lode-fissure filled with mullock. Showed no pyrite.

m. From the same cross-cut, 33 feet farther in; quartz-vein 2 inches wide. Showed no sulphides.

n Seven feet from the end of the same cross-cut. Broken country with numerous quartz-veins, showing sulphides freely.

o. From the end of the same cross-cut.

TABLE XIV.—*Concluded.*

Sample.	Distance from Auriferous Lode.	Locality.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.		Lbs.	Grains.	Grains.		Grains.	Grains.
p.....	3	Tipperary high-level adit.	4.48	211	81	0.047	0.0003	0.15
q.....	36	"	4.48	186	41	Trace.	Nil.	Nil.
r.....	12	"	4.48	542	360	0.58	0.0019	0.95
s.....	47	"	4.48	396	184	0.36	0.0027	1.35
t.....	1	"	4.48	216	74	0.019	0.0063	3.15
u.....	25	"	4.48	347	85	Trace.	Nil.	Nil.
v.....	46	"	4.48	616	422	0.70	"	"

p. From cross-cut No. 1. Solid country on foot-wall side; showed no pyrite.

q. From the same cross-cut, 33 feet farther in. Concentrates showed very little pyrite.

r. From No. 2 cross-cut on the hanging-wall side. Lode-fissure at this part filled with "mullock." Sample from broken country, showed a good deal of pyrite and arsenopyrite.

s. Sample showed a good deal of quartz, with a little pyrite.

t. From cross-cut No. 3; hanging-wall, broken country, opposite a part where the reef-fissure lay very flat; showed no sulphides.

u. From the same cross-cut; country still broken; showed no sulphides.

v. Sample taken from the end of cross-cut No. 3; showed a good proportion of sulphides. Concentrates nearly all sulphides.

TABLE XV.—*Analyses of Samples from the Deep Levels, Reefton Gold-Field.*

Sample.	Distance from Reef.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton.
	Feet.	lbs.	Grains.	Grains.	Grains.	Grains.	Grains.
a.....	From hanging-wall of auriferous reef in Progress Mine.	4.48	365 With a good percentage of sulphides.	218	0.36	0.013	6.5
b.....	Foot-wall of same reef.	4.48	704 Nearly all sulphides.	593	0.89	Nil.	Nil.
c.....	16 from same reef.	4.48	1248 Large percentage of sulphides.	975	1.83	0.010	5.
d.....	60 from same reef.	4.48	910	463	0.71	0.0005	0.25
e.....	140 from same reef.	4.48	471 Nearly all sulphides.	309	0.54	Nil.	Nil.
f.....	195 from same reef	4.48	243 No sulphides.	65	Trace.	"	"
g.....	200 from same reef.	4.48	1110 Nearly all sulphides.	873	1.80	"	"
h.....	Foot-wall of auriferous reef from the "Wealth of Nations."	4.48	176	74	0.16	0.0011	0.55
i.....	From same level, 10 feet farther from the reef.	4.48	904 Nearly all sulphides.	687	1.02	0.017	8.5
j.....	From same level, 58 feet farther from the reef.	4.48	703	489	0.73	0.0002	0.1
k.....	From same level, 90 feet farther from the reef.	4.48	217 No sulphides.	83	Trace.	Nil.	Nil.
l.....	From same level, 185 feet farther from the reef.	4.48	224	83	Nil.	"	"
m.....	From same level, 200 feet farther from the reef.	4.48	618	504	1.06	"	"

a. Soft-grained slate, much broken and showing many slickensides. The faces of the slickensides showed streaks of pyrite, due to crushing. *b*. Fine-grained slate, also much broken and contorted, with a number of small quartz-veins. Showed a large quantity of pyrite. *c*. Fine-grained slate, showing a large quantity of arsenopyrite and pyrite with a little galena. *d* and *e*. Solid slate, little altered, showing no signs of crushing Sulphides in fair quantity. *f*. Hard, fine-grained slate, showing no sulphides. *g*. Hard, fine-grained sandstone, showing a large quantity of pyrite. *h*. Fine-grained slate, very much broken, with seams of quartz. The surface of slickensides was coated with calcite. No sulphides visible. *i*. Fine-grained slate, contained large quantity of pyrite and arsenopyrite. *j*. Solid slate, with a vein of quartz about ½ inch wide. Sulphides visible in fair quantity. *k* Solid, very hard slate, no sign of breaking. Showed no sulphides. *l*. Solid slate, showing no sign of either quartz or pyrite. *m*. Solid slate, but showing pyrite and arsenopyrite freely.

TABLE XVI.—*Analyses of Samples of Soft Decomposed Andesite (Propylite) from Moanataiari Tunnel.*

Sample.	Approximate Distance from Auriferous Reef.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.	Yield of Silver.	Yield of Silver per Ton of Country-Rock.
	Feet.	Lbs.	Grs	Grains.		Grains.	Grains.	Grains.	Oz. dwt. grs.
A ...	50 From No. 9 Moanataiari Reef.	4.48	1246	716	1.03	0.0007	0.35	8.5	8.17.16
A'..	Ditto.	2.24	479	208	0.56	0.0006	0.6	3.34	6.19.4
									Grains.
B ...	55 From No. 9 Reef, close to a small leader 6 in. wide.	4.48	516	140	0.086	0.0013	0.65	0.0068	3.4
C....	Ditto.	4.48	1064	733	1.25	Nil.	Nil.	0.0163	8.15
D....	60 From No. 9; 20 ft. from a leader.	4.48	643	217	0.38	0.0003	0.15	0.0051	2.55
E....	65 From No. 9; foot-wall of a'rif's ld'r.	4.48	412	106	0.18	Nil.	Nil.	0.0017	0.85
F....		1000 grs.					345.0		151.
G ...	70 From No. 9; 20 ft. from a'rif's ld'r.	2.24	873	495	0.84	0.0012	1.2	0.019	19.
H...	70 From No. 9; 20 ft. from a flat qu'rz-seam, non-auriferous	4.48	317	90	0.062	Nil.	Nil.	0.0012	0.6
I....	20 From Don Pedro Re'f; 8 ft. from a slide dipping W. 70°	4.48	1132	795	1.32	0.0007	0.35	0.0237	11.85
J....	3 From Don Pedro Re'f.	4.48	375	209	0.44	Nil.	Nil.	0.0008	0.4

A. Grayish propylite, showing pyrite in good quantity, 20 feet from the Moanataiari fault. A'. Grayish propylite, showing pyrite in good quantity, 20 feet from the Moanataiari fault. B Light-brown propylite, showing very little pyrite; from the foot-wall of a small auriferous leader. Concentrates contained very little sulphides. C. White propylite, showing large quantity of pyrite; from the hanging-wall of a small auriferous leader. Concentrates contained good percentage of pyrite. D. Soft gray propylite, showing very little pyrite. Concentrates showed only a little pyrite. E. Hard bluish-gray propylite, foot-wall of a small auriferous quartz-vein, about 4 inches wide No pyrite visible. F. Quartz-vein, mentioned under "E," 4 inches wide. All assayed. G. Soft, light-brown propylite, from the vicinity of a "slide," dipping north 30°. The sample showed little pyrite. Concentrates contained good percentage of pyrite. H. Hard, gray propylite, showing no sulphides; from solid country, midway between two slides. Concentrates contained very little sulphides. I. White propylite, showing bluish patches, which gave the sample a mottled appearance. Plentifully impregnated with pyrite. Concentrates contained large percentage of pyrite. J. Propylite, brown from oxidation, which has probably gone on since the tunnel was opened. From the hanging-wall of Don Pedro vein; auriferous quartz, averaging about 10 inches wide. No sulphides visible in the sample.

TABLE XVI.—*Concluded.*

Sample.	Approximate Distance from Don Pedro Vein.	Weight Examined.	Weight of Concentrates Obtained.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.	Yield of Silver.	Yield of Silver per Ton of Country-Rock.
	Feet.	Grs.					Grains.		Grs.
K.....	From the vein.	1000	.....	.....	.....	.....	393	.....	177.
		Lbs.	Grs.	Grs.		Grains.	Grains.	Grains.	
L.....	1 to the foot-wall side	4.48	717	289	0.74	Nil.	Nil.	Nil.	Nil.
M.....	20; about 20 feet from the slide.	4.48	316	143	0.19	"	"	"	"
N.....	8	1.12	615	419	0.75	0.0019	3.8	0.0037	7.4
O.....	15	1.12	317	183	0.36	0.0036	7.2	0.0179	35.8
P.....	34	4.48	254	78	0.21	Nil.	Nil.	Nil.	Nil.
Q.....	65 East.	2.24	665	248	0.46	0.007	7.	0.0025	2.5
R.....	100	4.48	216	64	0 37	Nil.	Nil.	Nil.	Nil.
S.....	130	4.48	1032	719	1.03	0.0009	4.5	0.0087	4.35
T.....	150	2.24	317	194	0.46	Nil.	Nil.	Nil.	Nil.
U.....	110	4.48	604	473	0.83	0.0005	0.25	0.0019	0.95
Z.....	50	2.24	375	190	0.44	0.0013	1.3	0.0062	6.2
A1 ...	30	4.48	846	615	0.92	0.0009	0.45	0.0081	4.5
B1 ...	6	4.48	770	648	1.02	0.0018	0.9	0.0012	0.60
E1 ...	Beside the reef.	4.48	512	396	0.59	0.0032	1.6	0.0025	1.25

K. Quartz from Don Pedro vein, showing a good quantity of pyrite All assayed.

L. Hard, gray propylite, oxidized on outside. No sulphides visible

M. Light brown propylite from solid country between Don Pedro vein and the nearest slide Very little pyrite visible.

N. Gray propylite with black stringer ½ inch wide, from near the Don Pedro reef. Concentrates obtained showing a good quantity of pyrite.

O. White propylite from seam 3 inches wide. The propylite showed a little pyrite. Concentrates contained a good percentage of pyrite.

P. Hard, gray propylite from the vicinity of a black seam dipping north. No sulphides visible. Concentrates contained very little sulphides

Q. Mixture of white and bluish-white propylite, with a small quartz vein attached. No sulphides visible, Concentrates contained a fair percentage of pyrite.

R. Hard, bluish-white propylite in solid country. Concentrates contained very little sulphides.

S. White, soft propylite from near a fault-seam filled with clay, strongly impregnated with sulphides Concentrates nearly all sulphides.

T. Gray propylite from the hanging wall of a quartz reef 2 inches wide. A little pyrite visible in the sample. Concentrates contained a good proportion of pyrite.

U. Soft, whitish propylite near small slide. Good percentage of pyrite in sample. Concentrates contained a good deal of pyrite.

Z. Moderately hard, gray propylite, brown on outside, due to ferric oxide. Showed small quantity of pyrite. Concentrates contained good proportion of pyrite.

A1. Fairly hard, greenish propylite, white in the more decomposed parts. Showed pyrite freely. Concentrates contained good proportion of pyrite

B1. Soft, white propylite, showing large quantity of pyrite. Concentrates nearly all pyrite.

E1. Soft, white propylite, with dark veins of ferrous sulphate, as well as much pyrite. Concentrates nearly all pyrite.

TABLE XVII.—*Analyses of Samples of Hard Andesite from Two Belts of Unkindly Country in Moanataiari Tunnel, Thames.*

Sample.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.	Yield of Silver.	Yield of Silver per Ton of Country-Rock.
	Lbs.	Grains.	Grains.		Grains.	Grains.	Grains.	Grains.
*a**	4.48	342	53	Trace.	Nil.	Nil.	Nil.	Nil.
b	4.48	209	67	Nil.	"	"	"	"
c	4.48	530	239	0.38	"	"	0.0008	0.4
d	4.48	575	270	0.41	0.0003	0.15	0.0017	0.85
e	4.48	116	29	Nil.	Nil.	Nil.	Nil.	Nil.
f	4.48	279	84	"	"	"	"	"
g	4.48	438	378	0.68	0.0072	3.60	0.0039	1.95
h	4.48							
i	4.48							
j	4.48	347	74	0.057	Nil.	Nil.	Nil.	Nil.
k								
*l**	4.48	189	32	Nil.	"	"	"	"
m	4.48	237	75	"	"	"	"	"
n	4.48	375	47	"	"	"	"	"

a. Very dense dark-blue andesite; showed no sulphides. Concentrates contained no sulphides.

b. Very dense dark-blue andesite, from within 2 feet of a hard white quartz-vein Non-auriferous. Concentrates contained no sulphides.

c. Decomposed band of brownish andesite; from hanging-wall of same quartz-vein. Showed pyrite in very small quantity. Concentrates contained fair proportion of pyrite.

d. Decomposed band of brownish andesite; showed a little more pyrite. Concentrates contained fair proportion of pyrite.

e. Very hard blue andesite; showed no sign of pyrite. Concentrates contained no sulphides.

f. Andesite, slightly decomposed, varying from blue to brown; showed no sulphides. Concentrates contained no sulphides.

g Greenish andesite, soft and much decomposed; showed a good deal of pyrite in the most decomposed parts. Concentrates nearly all sulphides.

h. Greenish andesite, soft and much decomposed; showed a good deal of pyrite in the most decomposed parts. Concentrates from the sample met with an accident.

i. Greenish andesite, soft and much decomposed; showed a good deal of pyrite in the most decomposed parts. Concentrates from the sample met with an accident.

j. Much decomposed grayish-green andesite, quite white in patches, from near a leader of barren quartz. Sample showed no pyrite. Concentrates contained very little pyrite.

k. Much decomposed grayish-green andesite, quite white in patches, from near a leader of barren quartz. Sample showed no pyrite. Concentrates not analyzed.

l. Solid, very hard blue andesite, very little decomposed, and showing no sign of pyrite. Concentrates contained no sulphides.

m. Solid, very hard blue andesite, very little decomposed, and showing no sign of pyrite. Concentrates contained no sulphides.

n Solid, very hard blue andesite, very little decomposed, and showing no sign of pyrite. Concentrates contained no sulphides.

In the case of samples marked * the whole of the sample after concentration was assayed after the concentrates had been examined.

TABLE XVII.—*Concluded.*

Sample.	Weight Examined.	Weight of Concentrates.	Weight of Concentrates of Specific Gravity over 3.18.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.	Yield of Silver.	Yield of Silver per Ton of Country-Rock.
	Lbs.	Grains.	Grains.		Grains.	Grains.	Grains.	Grains.
o	4.48	304	103	0.19	Nil.	Nil.	0.0004	0.2
p	.....	.....	.....	.....	.....	.....	.....	.....
q	4.48	635	270	0.39	0.0005	0.25	0.0117	5.85
r	4.48	173	27	Nil.	Nil.	Nil.	Nil.	Nil.
s	4.48	219	80	0.016	"	"	"	"
*t**	4.48	346	73	Trace.		Button lost.	.....	.....
u	4.48	204	34	"	Nil.	Nil.	Nil.	Nil.

o. Greenish blue, slightly decomposed andesite, taken from near a small quartz-vein 1 inch wide, not auriferous. Sample showed a very little pyrite. Concentrates contained a little pyrite.

p. Greenish blue, slightly decomposed andesite, taken from near a small quartz-vein, 1 inch wide, not auriferous. Sample showed a very little pyrite Concentrates not analyzed.

q. Mottled blue and gray andesite, fairly hard, 20 feet northeast of the great Moanataiari fault. Sample showed a little pyrite in the more decomposed parts. Concentrates contained a fair quantity of pyrite.

r. Hard, blue andesite, very little decomposed. No sign of sulphides. Concentrates contained no sulphides.

s. Mottled gray and blue andesite, little decomposed. Showed very little sulphide. Concentrates contained no sulphides.

t. Hard, blue andesite, not at all decomposed. No sulphides. Concentrates contained no sulphides.

u. Hard, blue andesite, not at all decomposed. No sulphides. Concentrates contained no sulphides.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE XVIII.—*Analyses of Various Minerals Isolated from the Older Crystalline and Newer Eruptive Rocks of New Zealand, Victoria and Queensland.*

A. ROCKS OF NEW ZEALAND.

Sample.	Locality.	Weight Taken.	Mineral Isolated.	Weight of Mineral Isolated.	Method of Separation.	Sulphides.	Results.
		Gms.		Gms.			
1.....	From the "Staircase," on the Sutherland Falls road, Milford Sound, W. Coast of Otago.	200	Mica.	200	Picked by hand.	None.	No gold or silver found.
2.....	From the "Staircase," on the Sutherland Falls road, Milford Sound, W. Coast of Otago.	150	"	150	Picked by hand.	"	No gold or silver found.
3.....	Clinton Gorge, N.W. of Lake Te Anau, on Milford Sound road.	150	"	150	Picked by hand.	"	No gold or silver found.
4.....	Roaring Creek, Arthur River Valley, near Milford Sound.	100	"	100	Picked by hand.	"	No gold or silver found.
5.....	On the track between Milford Sound and Lake Ada, W. Coast of Otago.	100	"	100	Picked by hand.	"	No gold or silver found.
6.....	Dusky Sound, W Coast of Otago.	.....	Hornblende.	.....	Picked by hand.	See note below.	See note below.
7.....	Dusky Sound, W. Coast of Otago.	500	Hornblende.	113	Sons't's solution	None.	No gold or silver.
8.....	Milford Sound, W. Coast of Otago.	100	Hornblende.	100	Picked by hand.	"	No gold or silver.
9.....	Preservation Inlet, W. Coast of Otago.	100	Hornblende.	100	Picked by hand.	"	No gold or silver.
10.....	West Jacket Arm, W. Coast of Otago.	140	Hornblende.	140	Picked by hand.	"	No gold or silver.

1. Gneiss, containing veins of mica, about ½ inch wide, mixed with quartz. Several pounds' weight of mica were easily obtained. 2. Gneiss, containing veins of mica, about ½ inch wide, mixed with quartz. Several pounds' weight of mica were easily obtained. 3. Granite, with muscovite in large plates, easily picked out of sample. 4 Gneiss; contained veins of muscovite, from which pieces 2 inches square were easily picked out. 5. Gneiss; contained veins of muscovite, from which pieces 2 inches square were easily picked out. 6. Syenitic gneiss, consisting of quartz, feldspar and hornblende, with a little biotite, the hornblende in a seam about 2 inches wide. Pyrrhotite and chalcopyrite, with minute specks of native copper, associated with the hornblende. Five hundred grammes of the rock gave 53 grammes of sulphides, mostly pyrrhotite, which contained 0.0053 gramme of silver. 7. Syenite. 8. Syenitic gneiss. A vein of hornblende about 2 inches wide occupied the greater part of the sample analyzed. 9. Syenite porphyry, showing large patches and crystals of hornblende. 10. Syenite porphyry, showing large patches and crystals of hornblende.

TABLE XVIII.—*Continued.*

A. ROCKS OF NEW ZEALAND.—*Continued.*

Sample.	Locality.	Weight Taken.	Mineral Isolated.	Weight of Mineral Isolated.	Method of Separation.	Sulphides.	Results.
		Gms.		Grammes.			
11...	George Sound, W. Coast of Otago.	500				See note below.	See note below.
12...	George Sound, W. Coast of Otago.	200	Hornblende.	200	Picked by hand.	None.	No gold or silver.
13...	George Sound, W. Coast of Otago.	1000	Hornblende.		Picked by hand.	See note below.	See note below.
14...	Skippers Creek, Central Otago		Magnetite.	250	Magnet.	None.	No gold or silver.
15...	Shotover River. Central Otago.		Magnetite.	150	Magnet.	None.	No gold or silver.
16... Note 2	Shotover River, Central Otago.		Magnetite.	180	Magnet.	None.	No gold or silver.
17...	Portobello, Otago Peninsula, near Dunedin.					See note below.	See note below.
18...	Portobello, Otago Peninsula, near Dunedin.	500	.....	Hornblende, 57 Mica, 43	Sonstadt's Solution.	None.	No gold or silver.
19...	Portobello, Otago Peninsula, near Dunedin.	500	.....	Biotite, 38 Hornblende. 26	Sonstadt's Solution.	None.	No gold or silver.
20...	Near Dunedin.		Magnetite.	60	Magnet.	None.	No gold
21...	Water of Leith, near Dunedin.		Hornblende, 2.8 to 3. Augite, 3 to 3.2.	16 25	Klein's Solution.	None.	No gold or silver.
22...	Near Oamaru, Otago.		Magnetite.	80	Magnet.	None.	No gold or silver.
23...	Cape Foulwind, W. Coast of Nelson.		Mica.	200	Picked by hand.	None.	No gold or silver.
24...	Inangahua River, Reefton.	500	Mica.	84	Klein's Solution.	None.	No gold or silver
25...	Cape Foulwind quarries, W. Coast of Nelson, N. Z.	500	Mica.	43	Sonstadt's Solution.	None.	No gold or silver.

11. Syenitic gneiss, consist ng of quartz, feldspar and biotite, with patches of hornblende. Sample showed pyrrhotite and pyrite. Five hundred grammes gave 18.56 grammes of these minerals chiefly, but no gold or silver was found. 12. "Granite," a rock containing pink feldspar, quartz, muscovite and bands of hornblende. 13. "Granite," a rock containing orthoclase, quartz, muscovite and hornblende in veins. One thousand grammes gave 12.18 grammes pyrrhotite and arsenopyrite, which yielded no gold or silver. 14. Chlorite-schist, containing numerous octahedral crystals of magnetite. 15. Hornblende-schist, containing numerous octahedral crystals of magnetite. 16. Heavy "black sand," result of erosion of mica-schist. 17. "Diorite," consisting of triclinic feldspar, hornblende, biotite, magnetite and a little quartz. One thousand grammes gave 4.934 grammes of pyrite, containing 0.037 grain of gold and 0.0041 grain of silver. 18. Diorite, like No. 17. 19. Diorite, like No. 17. 20. Phonolite. 21. Basalt, very much decomposed, with large, undecomposed crystals of augite and hornblende. 22. Hard, solid basalt, from a quarry. 23. Gneiss-granite, with large folia of mica, easily picked out of sample. 24. Fine-grained granite. 25. Fine grained granite.

TABLE XVIII.—A. ROCKS OF NEW ZEALAND.—*Continued.*

Sample.	Locality.	Weight Taken.	Mineral Isolated.	Weight of Mineral Isolated.	Method of Separation.	Sulphides.	Results.
		Gms.		Gms.			
26...	Rangitoto, Westland, N.Z.				Klein's Solution.	See note below.	See note below.
27...	Bed of Hokitika river, Westland, N. Z.	500	Mica.	48	Klein's Solution.	None.	No gold or silver.
28...	Bligh Sound, W. Coast Otago.	500	"	61	Klein's Solution.	"	No gold or silver.
29...	Moanataiari tunnel, Thames district.	1000	Pyroxenes of sp. gr. 2.9 to 3.2.	73	Klein's Solution.	"	No gold or silver.
30...	Moanataiari tunnel, Thames district.	1000	Hornblende and chlorite between sp. gr. 2.8 and 3.1.	56	Klein's Solution.	"	No gold or silver.
31...	Waitekauri creek, Thames district.	1000	Hornblende and chlorite between sp. gr. 2.8 and 3.1.	65	Klein's Solution.	"	No gold or silver.
32...	Moanataiari tunnel, Thames district.	1000	Pyroxen's between sp. gr. 2.9 and 3.2.	43	Klein's Solution	"	No gold or silver.
33...	Fame and Fortune, G. M. Co., Thames.	1000	Pyroxen's between sp. gr. 2.9 and 3.2.	84	Klein's Solution.	"	No gold or silver.
34...	Fame and Fortune G. M. Co., Thames.	500	Pyroxen's between sp. gr. 2.9 and 3.2.	27	Klein's Solution.	"	No gold or silver.
	B. ROCKS OF VICTORIA.						
35...	900 ft. level Long Tunnel G. M. Co.	1000	Hornblende.	73	Klein's Solution.	None.	No gold or silver.
36...	Thomson river, ne'r Walhalla, Gippsland, Vict.	1000				See note below.	See note below.
37...	Thomson river, ne'r Walhalla, Gippsland, Vict.	1000	Hornblende.	38	Klein's Solution.	None.	No gold or silver.
38...	Big Hill, Bendigo, near the Silurian boundary.		"			See note below.	See note below.
39...	Big Hill, Bendigo, near the Silurian boundary.	500	Mica.	53	Klein's Solution.	None.	No gold or silver.
40...	Big Hill, Bendigo, near the Silurian boundary.	500	"	84	Klein's Solution.	"	No gold or silver.
41...	From Gabo Island, S. E. of Vict., used as a building stone in Melbourne.	1000	Hornblende.	96	Klein's Solution.	"	No gold or silver.
42...	Ballarat East, Victoria.		Magnetite.	50	Magnet.	"	No gold or silver.
43...	Sunbury, near Melbourne, Victoria.		"	40	"	"	No gold or silver.

26. Granite, with a pyritous vein. One thousand grammes gave 28.92 grammes of arsenopyrite and pyrrhotite, with a little galena, and this yielded 0.0082 grain of gold and 0.0019 grain of silver.
27. Granite.
28. Gneiss.
29. Nepheline-andesite, hard, very little decomposed.
30. Hard blue hornblende-andesite.
31. Hornblende-andesite, slightly altered.
32. Augite-andesite, with much chlorite.
33. Hard blue unaltered augite-andesite.
34. Hard blue unaltered augite-andesite.
35. "Diorite" dike, near auriferous reef, Walhalla, Gippsland, Victoria.
36. "Diorite," associated with copper-lode. Showed chalcopyrite, of which there were obtained from 1000 grammes of the rock 16.48 grammes (with a small quantity of bournonite), yielding 0.407 grain of silver and no gold.
37. "Diorite," like No. 36.
38. Syenitic granite (quartz, orthoclase and muscovite, with large crystals of hornblende). One thousand grammes gave 16.03 grammes of pyrite, yielding neither gold nor silver.
39. Syenitic granite, like No. 38.
40. Granite.
41. Syenite.
42. Very dense basalt, overlying auriferous slates.
43. Vesicular basalt.

TABLE XVIII.—C. ROCKS OF QUEENSLAND.—*Concluded.*

Sample.	Locality.	Weight Taken.	Mineral Isolated.	Weight of Mineral Isolated.	Method of Separation.	Sulphides.	Results.
		Gms.		Gms.			
44...	The 1070-ft. level, Brilliant & St. George G. M. Co.. Charters Towers, Queensland.	500	Sp. gr. 2.75 to 3 (biotite). Sp. gr. 3 to 3.2 (hornblende).	21 50	Klein's Solution.	None.	No gold or silver.
45....	The 1070-ft. level, Brilliant and St. George G. M. Co., Charters Towers, Queensland.	500	Sp. gr. 2.75 to 3 (biotite). Sp. gr. 3 to 3.2 (hornblende).	47 40	Klein's Solution.	"	No gold or silver.
46....	The 950-ft. level of the No. 5 North Phœnix G. M. Co., Gympie.	1000		36		See note below.	See note below.
47....	The 950-ft. level of the No. 5 North Phœnix G. M. Co., Gympie.	1000		25		See note below.	See note below.

44. Quartz-mica diorite (tonalite: the "granite" of Charters Towers).

45. Quartz mica diorite (tonalite: the "granite" of Charters Towers).

46. Diorite-aphanite, with much carbonate of lime (calc-aphanite); a very much altered rock; effervesces strongly with acid (the "greenstone" of the Gympie gold-field). It contains pyrite and sphalerite in small quantities. One thousand grammes gave 8.42 grammes of pyrite, yielding no gold or silver.

47. Diorite-aphanite, with much carbonate of lime (calc-aphanite); a very much altered rock; effervesces strongly with acid (the "greenstone" of the Gympie gold-field). One thousand grammes gave 42.7 grammes of pyrite and galena, which yielded 0.0016 grain of gold and 0.0037 grain of silver.

TABLE XIX.—*Examination of Mine-Waters of the Vadose Region for Free Acid and Ferric Salts.*

Sample.	Country-Rock.	Reaction to Test-Paper.	Free Acid in Grammes per Liter before Exposure to Air.		Free Acid in Grammes per Liter after Exposure to Air.		Total Weight of Iron in Grammes per Liter.	Iron Present as Ferric Salts.	Iron Present as Ferrous Salts.
			HCl.	H_2SO_4.	HCl.	H_2SO_4.			
*a**...	Prop'lite highly pyritous.	Strongly acid.	0.446	7.901	0.446	8.650	3.198	2.431	0.767
*b**...	Prop'lite highly pyritous.	Strongly acid.	0.287	Nil.	0.592	1.382	2.471	0.946	1.525
*c**..	Prop'lite highly pyritous.	Acid.	0.065	"	0.208	Nil.	1.036	0.731	0.355
*d**..	Prop'lite highly pyritous.	Strongly acid.	0.506	6.078	0.506	8.921	2.017	0.896	1.111
*e**...	Prop'lite highly pyritous.	Acid.	0.079	Nil.	0.361	Nil.	0.968	0.834	0.134
*f**...	Propylite and rhyolite.	"	0.216	"	0.465	0.380	0.758	0.210	0.548
*g**..	Propylite, pyritous.	"	0.409	1.063	0.605	1.582	1.903	1.127	0.776
h....	Mica-schist.	Neutral.	Nil.	Nil.	Nil.	Nil.	0.095	0.027	0.068
i....	Mica-schist.	"	"	"	"	"	0.872	0.305	0.567
j....	Slate and sandstone.	Highly acid.	"	"	"	"	0.569	0.481	0.088
k....	Mica schist.	Highly acid.	0.087	"	0.105	"	1.007	0.783	0.224
l....	"	Strongly acid.	0.658	"	0.816	"	1.569	1.406	0.163
m...	"	Slightly acid.	0.064	"	Nil.	"	0.987	0.639	0.348
n....	"	Neutral.	Nil.	"	"	"	0.640	0.379	0.261
o....	"	Acid.	"	"	0.406	"	1.206	0 217	0.989
p.....	Slate and sandstone.	Neutral.	"	"	Nil.	"	0.489	0.426	0.063
q....	Slate and sandstone.	Neutral.	"	"	"	"	0.602	0.578	0.024

a. From the Whau mine, Thames, N. Z. Contained a large quantity of iron in solution. Color, wine-red. Sp. gr., 1.021.

b. From Maria reef, Karangahake, Thames, N. Z. Deposited a large quantity of ferric hydroxide on standing.

c. From Woodstock reef, Thames, N. Z. Behaved on standing like sample *b*.

d From the Alburnia mine, Thames, N. Z. In appearance like sample *a*. Sp. gr., 1.019.

e From the Grace Darling mine, Thames, N. Z. Nearly clear; slight deposit of ferric hydroxide on standing.

f. From the Martha mine, Waihi, Thames, N. Z. Appearance and behavior on standing like sample *e*.

g. From the Crown mine, Karangahake, Thames, N. Z. Like samples *e* and *f*.

h. From the Tipperary mine, Macetown, Otago. Clear.

i. From the Premier mine, Macetown, Otago. Clear.

j. From the Long Tunnel, Walhalla, Victoria. Clear.

k. From the Bonanza mine, Nenthorn, Otago. Clear.

l. From a quartz-reef near Roxburgh, Otago. Reddish; deposited a good quantity of ferric hydroxide on standing.

m. From the Bella reef, Waipori, Otago. Clear.

n. From the Gabriel's Gully reef, Lawrence, Otago. Clear.

o. From the Game Hen reef, Hindon, Otago. Clear.

p. From the reef on Sovereign Hill, Ballarat, Victoria. Clear.

q. From the reef on Big Hill, Bendigo, Victoria. Clear.

TABLE XX.—*Analyses of Country-Rock from the Vadose Region of Walhalla Gold-Field, Gippsland, Victoria.*

Sample.	Approximate Distance from Auriferous Reef (Cohen's Reef).	Weight Examined.	Weight of Concentrates Obtained.	Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Pounds.	Grains.		Grains.	Grains.
a	3 West.	4.48	1904 Chiefly oxide of iron, showing a little mica.	Trace.	Nil.	Nil.
b	10 "	4.48	1650	"	"	"
c	16 "	2.24	1824	Nil.	0.027	27.
*d**	100 "	2.24	All assayed.	"	Nil.	Nil.
e	240 "	2.24	1463	"	0.0183	18.3
f	360 "	1000 grs.	All assayed.	"	Nil.	Nil.
g	540 "	4.48	1230 Chiefly quartz, ferric oxide, with a little mica.	"	0.0027	1.35
h	1200 "	2.24	1640	"	0.0076	7.6
i	6 East.	2.24	1356 Mostly ferric oxide, with a little magnetite.	"	0.029	29.
*j**	75 "	2.24	All assayed.	Trace.	0.0017	1.7
k	360 "	2.24	1748 Chiefly ferric oxide.	Nil.	Nil.	Nil.

a. Very hard, coarse-grained, solid sandstone, much stained with oxide of iron.

b. Fine-grained, hard, solid slate, very little altered in any respect.

c. Oxide of iron from a cavity between sandstone and slate. The country much broken near where sample was taken.

d Slate, fine-grained, very little altered, very solid and hard.

e. Broken sandstone, slate and oxide of iron, from a small fault in the rock, exposed to the surface.

f. White quartz, slightly iron-stained, from small reef-formation about 4 inches wide.

g. Hard, fairly coarse-grained sandstone, with seams of ferric oxide along the bedding-planes.

h. Slate. sandstone and ferric oxide from broken country, filling a slight fault.

i Ferric oxide, coating slate and sandstone.

j. Hard, solid, fine grained slate, little altered.

k. Hard sandstone, much stained with ferric oxide

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examinad.

TABLE XXI.—*Analyses of Country-Rock from the Vadose Region of the Ballarat Gold-Field, Victoria.*

Sample.	Approximate Distance from Auriferous Reef.	Weight Examined.	Weight of Concentrates Obtained.		Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Lbs.	Grains.			Grains.	Grains
a	8 East.	1.12	684	Mostly ferric oxide and quartz.	Nil.	0.037	74
b	60	2.24	1050		"	0.017	17
c	140	4.48	1812		"	0.0063	3.15
d	186	4.48	1485		"	0.0023	1.15
e	240	4.48	2116		"	0.0016	.8
f	360	4.48	1642		"	0.0058	2.9
g	500	1.12	856		"	Nil.	Nil.

a. Oxide of iron, with a little quartz and loose sand, formed by the disintegration of sandstone.

b. Oxide of iron, from a joint separating two adjacent strata of sandstone.

c. Soft sandstone, pure white.

d. White pipe-clay, with yellow streaks. A product of the decomposition of slate.

e. Soft, fine-grained pipe clay, colored from red to purple by oxide of iron.

f. Oxide of iron, mixed with soft sandstone and clay from broken country.

g. Oxide of iron, forming clay parting between two beds of pipe-clay.

TABLE XXII.—*Analyses of Rocks from the Vadose Region of Otago, New Zealand.*

Sample.	Distance from Auriferous Reef.	Approx. Depth Below Surface.	Locality.	Weight Taken.	Weight of Concentrates.		Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Ft.		Lbs.	Grains.			Grains.	Grs.
a.....	4	40	From a tunnel driven along the Bella reef, Waipori, Otago.	4.48	643	Mostly mica and oxide of iron.	Nil.	0.0016	0.8
b.....	20	40	From a tunnel driven along the Bella reef, Waipori, Otago.	4.48	1246		"	0.0107	5.35
c.....	3	50	From a tunnel driven along the Bella reef, Waipori, Otago.	4.48	1675		"	0.065	32.55
d.....	60	30	Near the mouth of the low-level tunnel, Tipperary G. M. Co., Macetown, Otago.	4.48	1217		"	Nil.	Nil.
e.....	35	60	From the same tunnel, but 25 feet nearer the reef.	2.24	1176		Trace.	0.0179	17.9
f.....	20	70	From the same tunnel, 15 feet nearer the reef.	2.24	1030		"	0.0086	8.6
g.....	160	35	From near the mouth of the low-level tunnel, Premier G. M. Co., Macetown, Otago.	1.12	774		Nil.	0.0018	3.6
h.....	90	85	From the same tunnel, but nearer the reef and at a greater depth.	2.24	617		Trace.	Nil.	Nil.
i.....	65	95	From the same tunnel, but 15 feet nearer the reef.	1.12	612		Nil.	0.0013	2.6
j.....	6	40	From the Bonanza reef, Nenthorn, Otago, within 6 feet of a rich auriferous reef.	2.24	1738	Principally mica and ferric oxide.	0.027	0.0116	11.6
k.....	3	40	From the same mine, at the same depth, but nearer the reef.	2.24	1247		Nil.	0.0073	7.3
l.....	600	20	From a cliff west of Saddle Hill reef, near Dunedin.	1.12	726		0.008	Nil.	Nil.
m.....	200	20	From the same locality, but nearer reef.	2.24	1073		Nil.	0.0005	0.5

a. Mica-schist, yellowish color, soft, very much decomposed.

b. Mica-schist, yellowish color, soft, very much decomposed.

c. Mica-schist, yellowish color, soft, very much decomposed.

d. Hard mica-schist, with quartz-bands ½ inch wide. Specimen much stained with oxide of iron.

e. Ferric oxide from a slight fault in mica-schist, showing a little quartz in veins.

f. Quartz and ferric oxide from broken country.

g. Solid mica-schist, much stained by ferric oxide.

h. Ferric oxide and broken mica-schist, from a small fault.

i. Solid mica-schist, much discolored by ferric oxide.

j. Much decomposed, very soft mica-schist, stained brown with ferric oxide.

k. Much decomposed, very soft mica-schist, stained brown with ferric oxide.

l. Ferric oxide filling the space in joint-planes of phyllite, which forms the country-rock of this reef.

m. Decomposed phyllite, much stained with ferric oxide.

TABLE XXIII.—*Analyses of Samples from Skipper's Road, near Lake Wakatipu, Otago.*

Sample.	Distance from Auriferous Reef.	Approx. Depth Below Surface.	Locality.	Weight Taken.	Weight of Concentrates.		Percentage of Sulphur.	Yield of Gold.	Yield of Gold per Ton of Country-Rock.
	Feet.	Feet.		Lbs.	Grains.			Grs.	Grs.
a.....	At least 2 or 3 miles.	50	All these samples were taken from a deep cutting on the Skippers' road.	2.24	1127	Chiefly ferric oxide and mica.	Nil.	Nil.	Nil.
b.....	About ¼-mile from *a*.	80		2.24	1416	With a little pyrite.	"	"	"
c.....	At least 1 mile.	30		2.24	1723		"	"	"
d.....	About 1000 yards from *c*.			2.24	1093	With a good percent'g of sulph'd's	1.384	"	"
e.....	From near sample *d*.	40		2.24	834	Showed no p'rite.	Nil.	"	"
f.....	About 3 ml's.	20		2.24	1568		"	"	"
g.....	From near sample *f*.	40		2.24	817		Trace.	"	"
h.....	Over 2 miles.	10		1.12	716		Nil.	"	"
*i**.....	Near samp. *h*.	15		1.12	618		"	"	"

a. Ferric oxide from joint-planes.

b. Broken rock; quartz and mica, with much contorted mica-schist, filling a fissure 18 inches wide.

c. Broken rock; quartz and mica.

d. Broken rock, but the sample contained a few large crystals of pyrite, somewhat decomposed to ferric oxide.

e. Broken rock, but showed no pyrite.

f. Quartz and broken mica-schist and ferric oxide, from a fault-fissure or lode-formation, about 1 foot wide, in mica-schist.

g. Quartz and broken mica-schist.

h. Ferric oxide, filling joint-planes in broken mica-schist.

i. Quartz-interlaminations, about 2 inches wide, in mica-schist, much stained with ferric oxide.

In the case of samples marked * the whole of the sample left after concentration was assayed after the concentrates had been examined.

TABLE XXIV.—*Analyses of Samples from the Vadose Region of the Ohinemuri District, Thames, New Zealand.*

Sample.	Depth Below Surface.	Approx. Distance from Reef.	Locality.	Weight Examined.	Weight of Concentrates.	Yield of Gold.	Yield of Gold per Ton.	Yield of Silver.	Yield of Silver per Ton.
	Feet.	Feet.		Lbs.	Grains.	Grains.	Grs.	Grains.	Grs.
a	10	30	From a cliff on the tramway between the Crown mine and Karangahake, Thames.	4.48	1634	0.0014	0.7	0.0018	0.9
b	20	180	From a fault.	1.12	All assayed.	0.009	18	0.0036	7.2
c	20	350	" "	4.48	1065 With a good percent'ge of pyrite.	0.0062	.10	0.0054	2.7
d.....	30	120	East of the Maria reef, Karangahake, Thames.	4.48	873 With a good proportion of pyrite.	Nil.	Nil.	Nil.	Nil.
e	80	3	Great Woodstock Tunnel, Karangahake, Thms.	1.12	All assayed.	0.0095	19	0.0034	6.8
f.....	10	60	From a cliff on the Waitawheta river, near the Crown mine, Thames.	2.24	" "	0.0013	1.3	0.0042	4.2
g.....	60	24	Near the Grace Darling reef, Waitekauri, Thames.	4.48	1372	0.0063	3.15	0.0048	2.4
h.....	60	30	Foot-wall side of Crown reef.	4.48	846 With a large percentage of sulphides.	Nil.	Nil.	Nil.	Nil.
i.....	25	600	200 yds. off hanging-wall side of the Crown reef, 10 ft. above the Waitawheta river.	2.24	617 Nearly all pyrite.	0.0093	9.3	0.0126	12.6
j.....	10	660	From a fault exposed on the tramway between the Crown mine and Karangahake, Thames.	1.12	All assayed.	0.0071	14.2	0.0014	2.8

a. Nearly white propylite; showed a good percentage of pyrite.

b. Ferric oxide and higher oxides of manganese, with a little quartz.

c. Solid, hard andesite, oxidized to brown color on outside. Showed a good deal of pyrite.

d. Hard, greenish hypersthene-andesite. Showed a good deal of pyrite.

e. Ferric oxide and higher oxides of manganese, from a vein near the foot-wall of the Great Woodstock reef.

f. Andesite; brown, much decomposed. No pyrite visible.

g Decomposed andesite and ferric oxide, from a fault.

h. Grayish white andesite, much oxidized on the outside, but showing pyrite freely when broken.

i. Nearly white, very siliceous rhyolite. Showing pyrite freely.

j. Ferric oxide and higher oxides of manganese.

Diagram 1. (See Table I.)

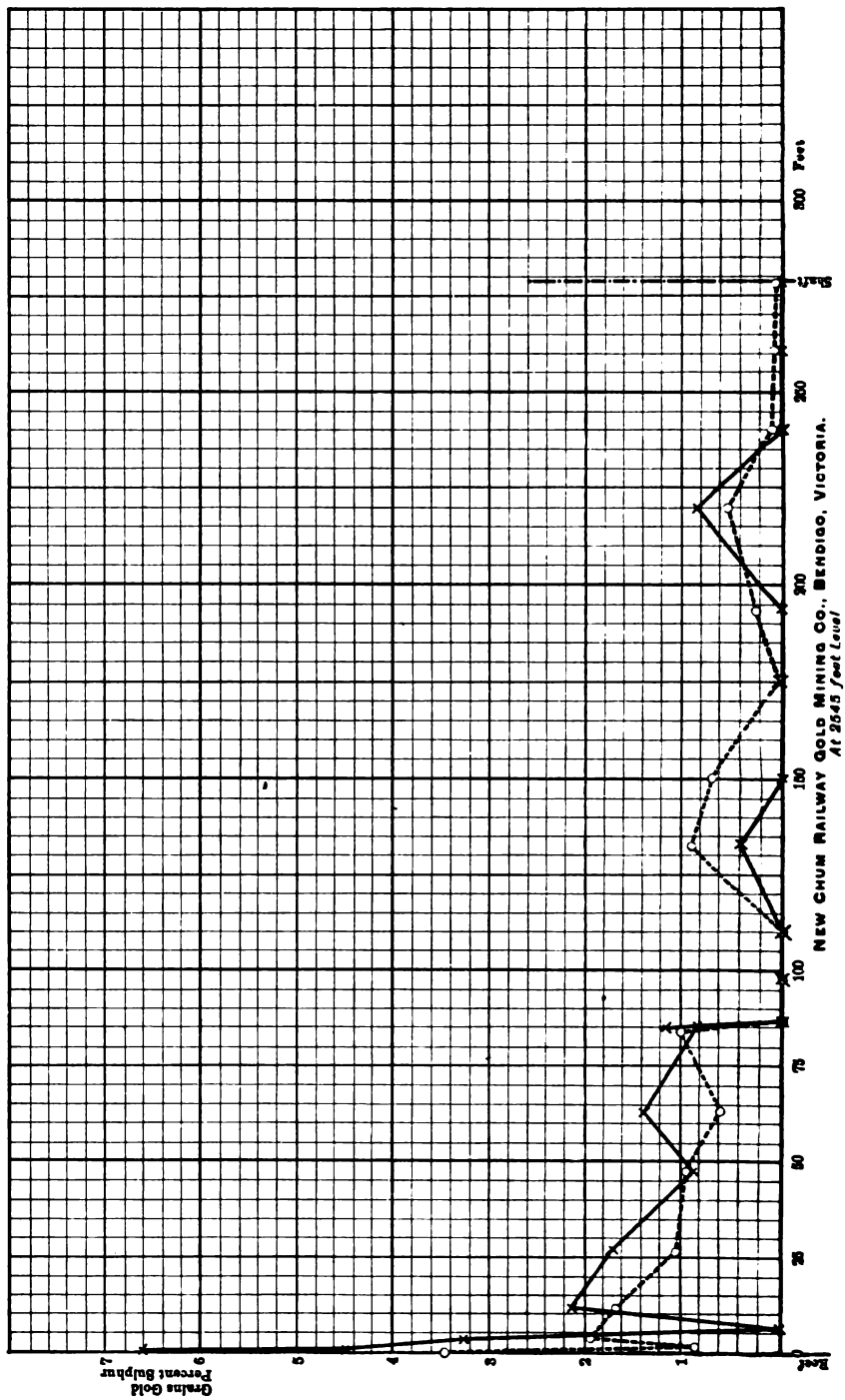
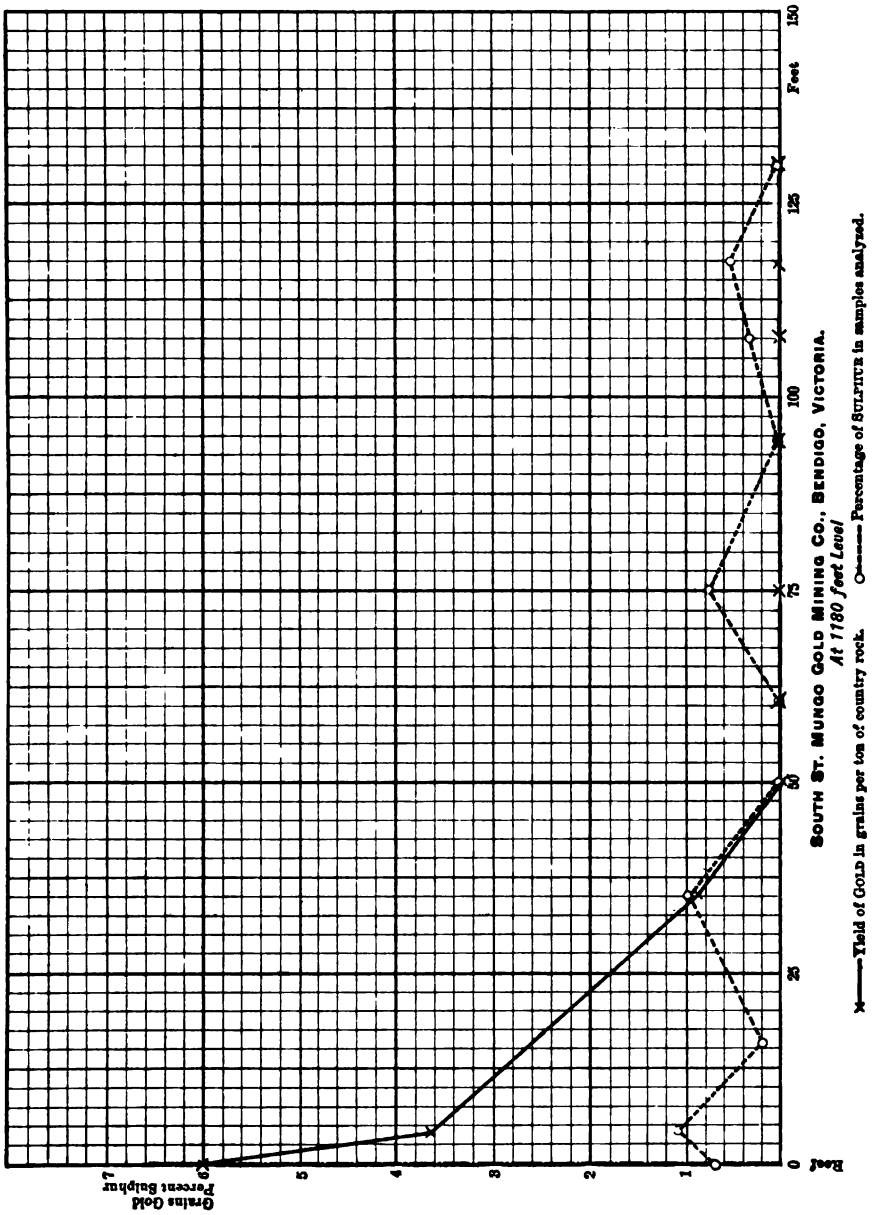


Diagram 2. (See Table II.)



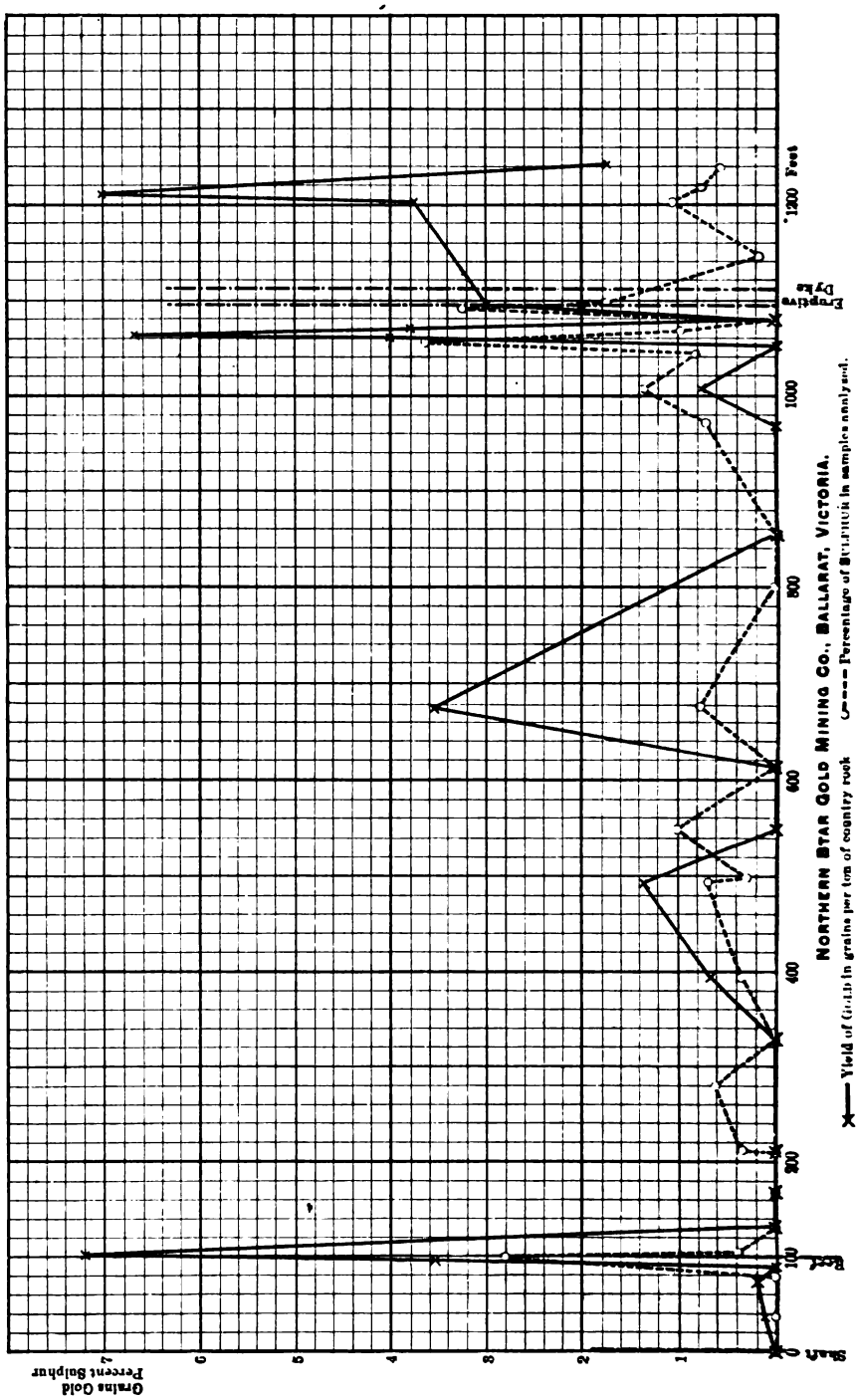


Diagram 3. (See Table V.)

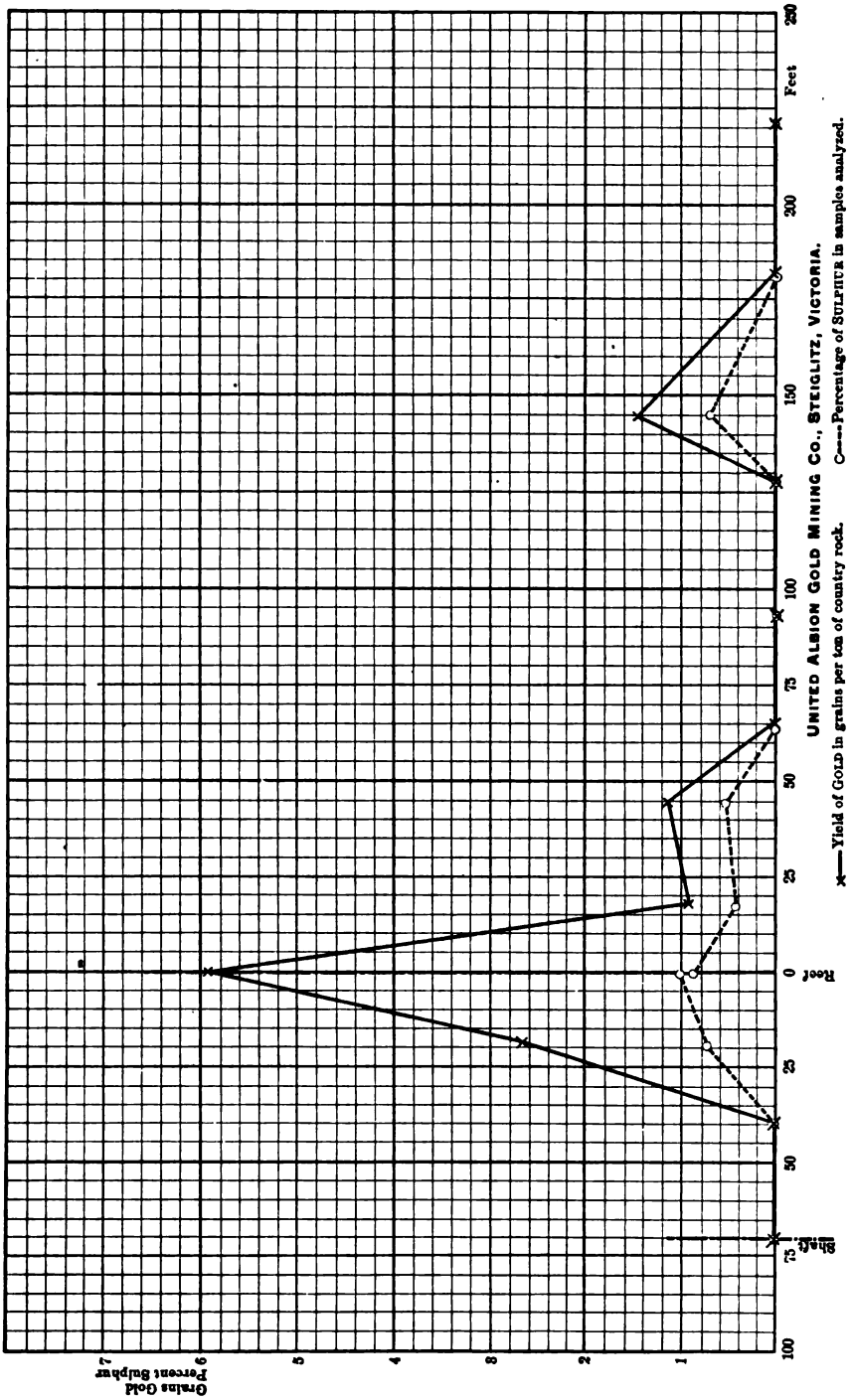


Diagram 4. (See Table VI.)

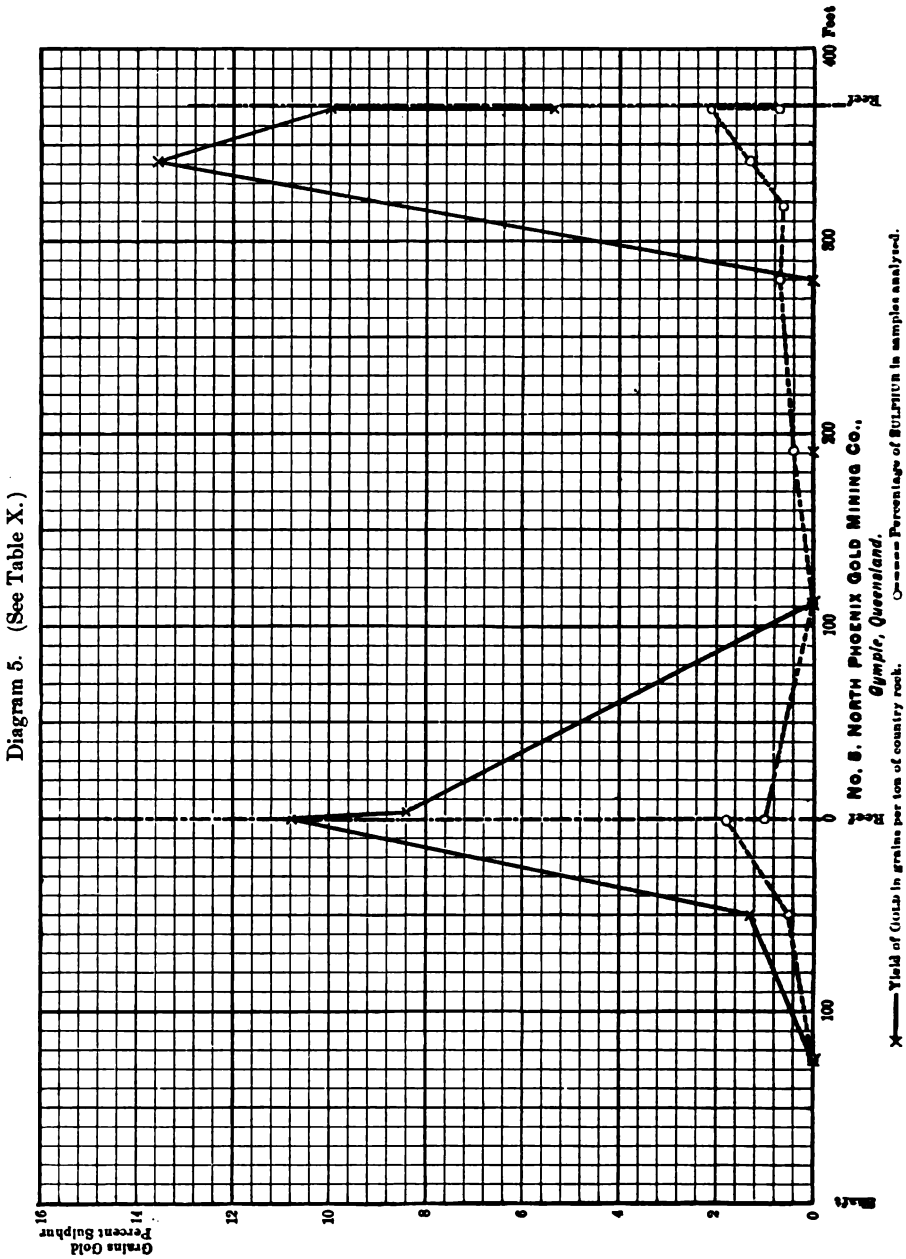


Diagram 5. (See Table X.)

Diagram 6. (See Table XIII.)

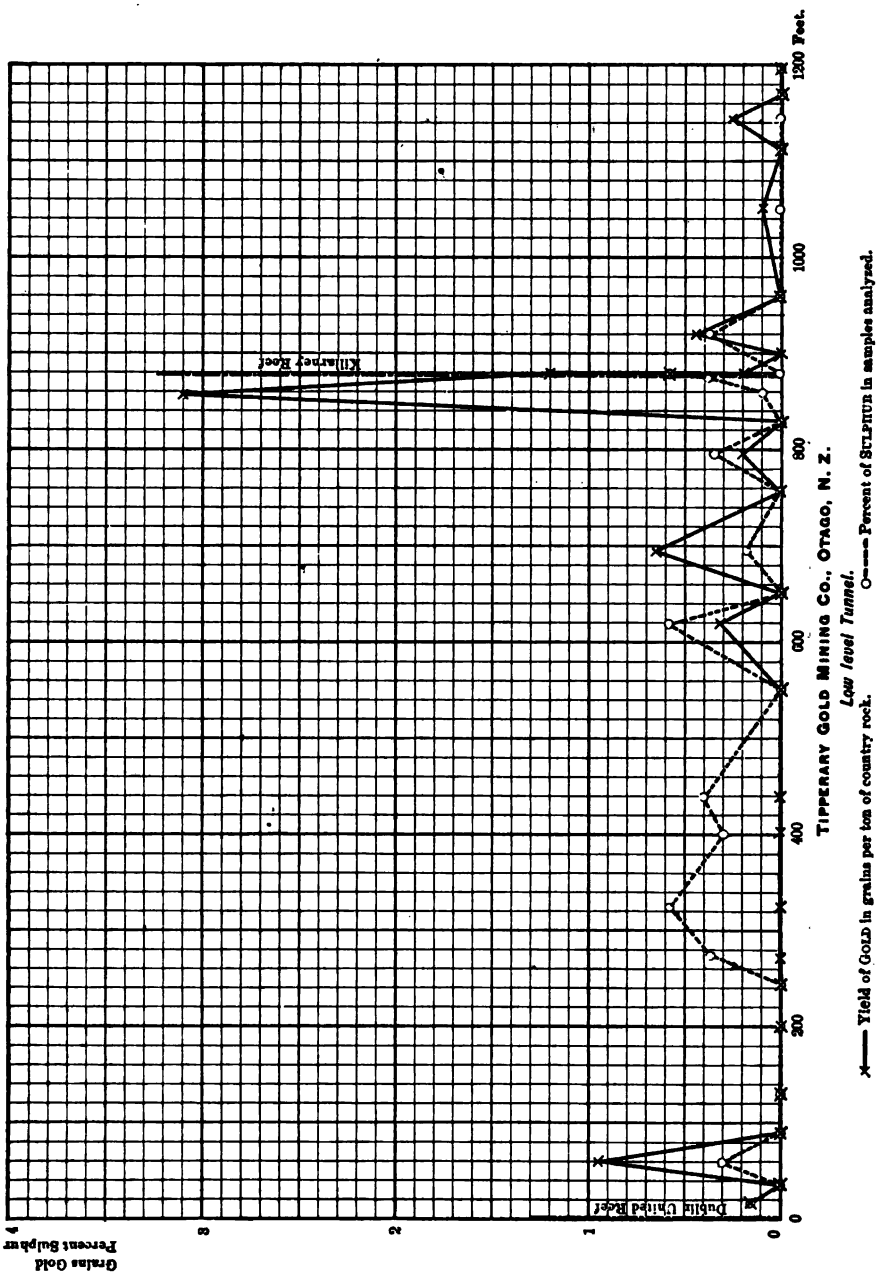


Diagram 7. (See Table XIV.)

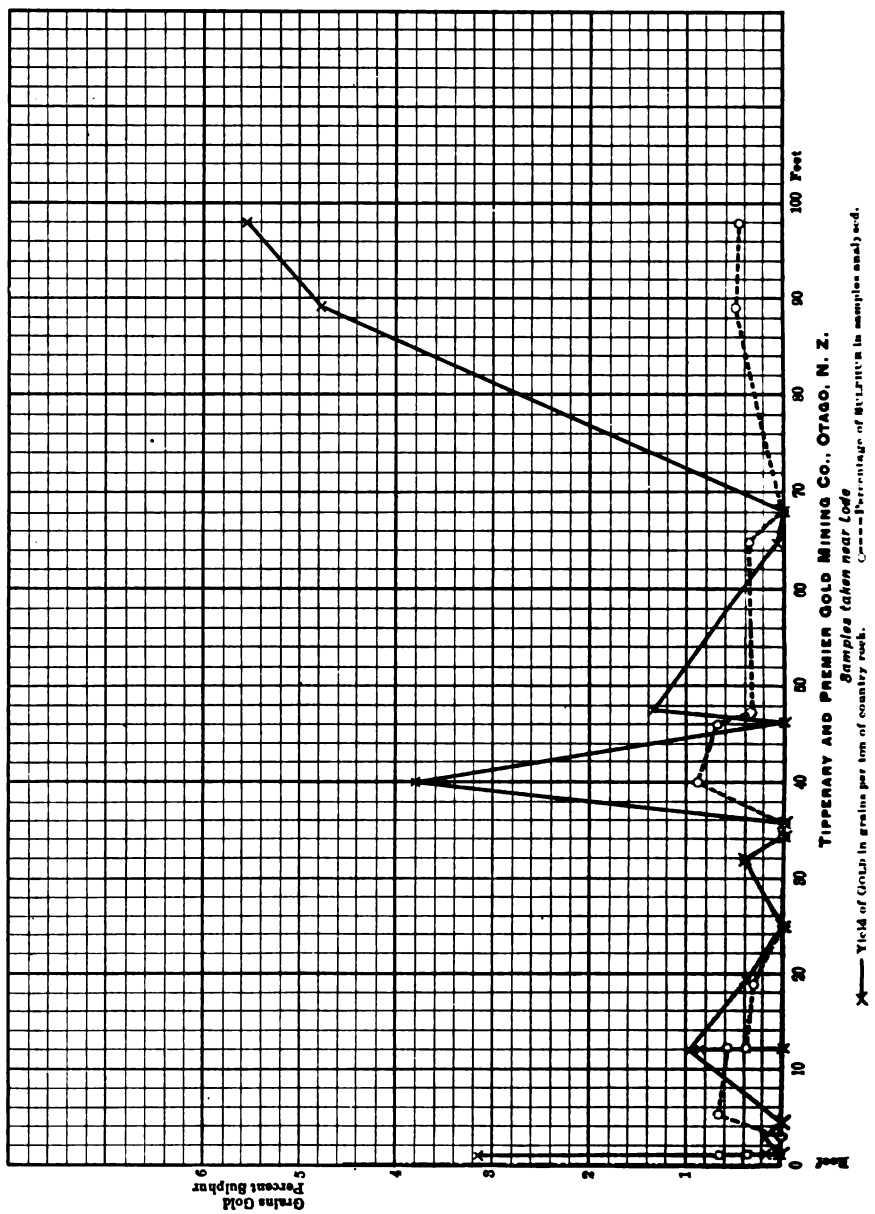


Diagram 8. (See Tables XX. and XXI.)

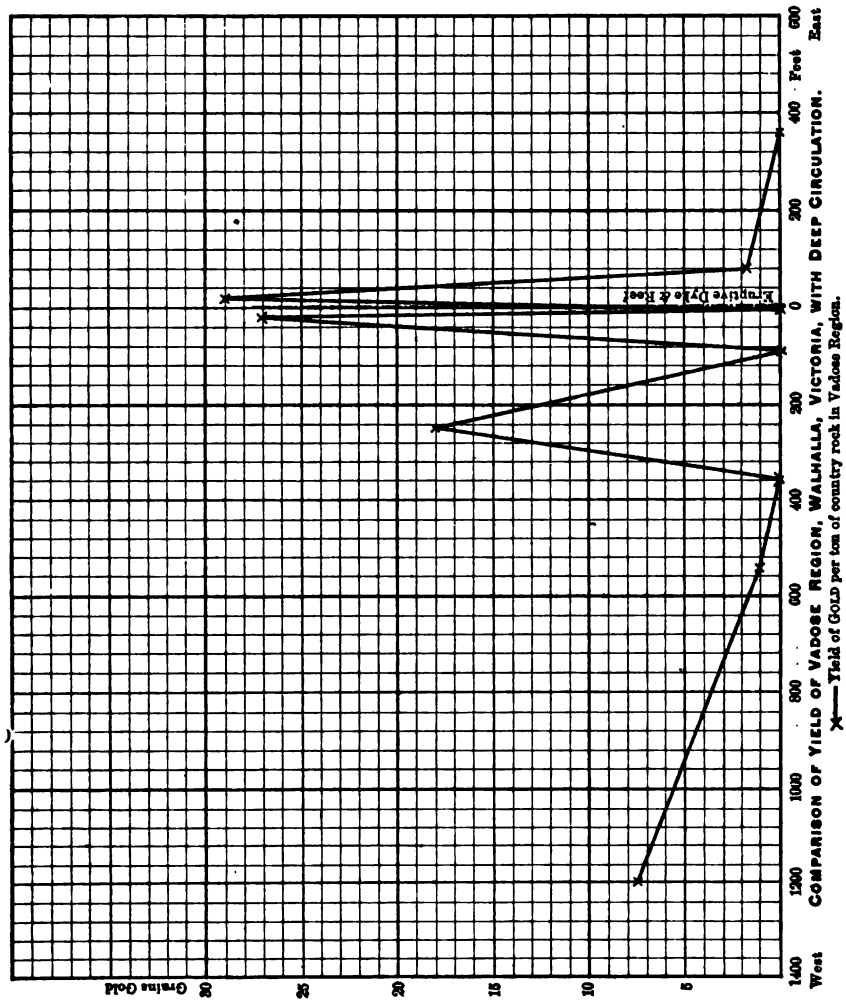
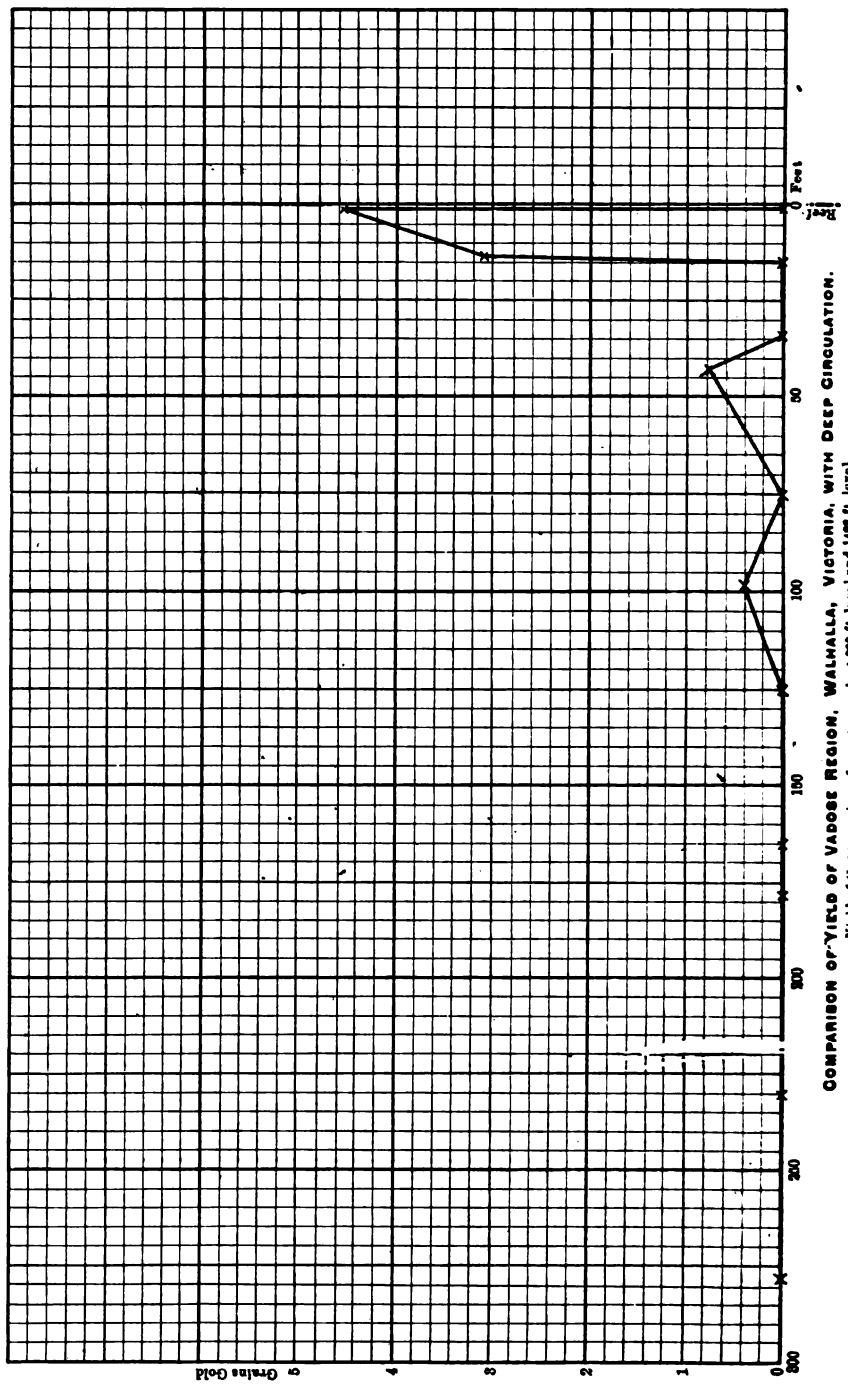


Diagram 9. (See Tables XX. and XXI.)



COMPARISON OF YIELD OF VADOSE REGION, WALHALLA, VICTORIA, WITH DEEP CIRCULATION.

Yield of GOLD per ton of country rock at 900 ft. level and 1488 ft. level.

The Origin and Mode of Occurrence of the Lake Superior Copper-Deposits.

BY DR. M. E. WADSWORTH,* HOUGHTON, MICHIGAN.

(Lake Superior Meeting, July, 1897.)

THE region about the south shore of Lake Superior is to geologists one of the most interesting districts of the United States, embracing as it does, in a limited area, old crystalline rocks, together with forms that are almost in their original condition of a beach sand and mud. In this region was first established the base of the geological column, the Azoic ("without life") System of Foster and Whitney, or the Archæan ("the beginning") of Dana. Overlying this system are found the sandstones and limestones of the Palæozoic ("ancient life"), and their associated lava-flows (Algonkian?).

These systems possess a strong economic interest, owing to the stores of iron in the Azoic and of copper in the Palæozoic or Eozoic of this district, which forms one of the most important mining regions in America.

The geology of this section is so difficult and complicated that, in its general discussion, perhaps no proposition can be stated concerning any portion of it to which exceptions cannot be taken. Indeed, out of the general discussion of different points comes in time the truth, and various geologists, even now, are working over this region in the endeavor to arrive at some consensus, or at least to determine upon what points they can agree, and upon what points difference of opinion will have to exist between them at present, until further evidence can be obtained. The writer will endeavor to give in a brief form that which appears to him at present to be the most correct statement of the geological structure of the region, admitting that from time to time, as more complete evidence shall be obtained, he expects to change his views in the future, as he has done in the past, if that evidence shall cause him to believe that he has been mistaken.

* President of the Michigan College of Mines.

The rocks south of Lake Superior have been formed in three ways: 1st, by mechanical means; 2d, by eruptive, igneous, or volcanic agencies; 3d, by chemical action.

Fragmental or Detrital Rocks.

The mechanical agencies of the Azoic time acted upon some prior-formed rocks in like manner as we see rain, winds, waves, frosts, etc., now breaking down the rocks of the present day, causing them to be deposited as soil, mud, sand and shingle, forming detrital or sedimentary deposits. Such detritus one can see upon the shores of any lake or sea, being in many localities variable in its composition, and oftentimes abruptly changing from fine mud to sand, or even to coarse shingle. At other localities upon the same lake shore one may observe a nearly uniform sand, mud or shingle stretching away as far as the eye can reach along that shore. Like uniformity and like abrupt changes are seen by the geologist in the rocks formed from the ancient muds, sands and shingles of the early seas and lakes. These deposits may have remained on the surface of the ancient beach, or may have been deeply buried under succeeding deposits; but whatever may have been their position relative to the earth's surface, they have been greatly changed or altered from their original condition, although the evidences of that original condition remain plainly visible to him who has learned to read the worn, torn and worm-eaten book of Nature. In truth it may be said that no event can take place without leaving its effects behind, and these can be interpreted with greater or less clearness until their record has been entirely obliterated.

To return: we find that these old muds, sands and shingles have been acted upon, and metamorphosed or altered, by heat from the earth's molten interior, or from contact with igneous or volcanic rocks, with their accompanying hot waters. Or, again, these deposits have been affected by hot or cold waters percolating through them, bearing materials which chemically act upon them; or, again, they may have been subjected to greater or less squeezing and pressure during the formation of the numerous wrinkles that old Mother Earth now wears upon her rugged face, deeply furrowed by her tears.

Of all the agents of consolidation and change in rocks, the chemically active waters are, to my mind, the most potent; and it appears to me probable that dry heat and pressure alone would be unable to produce any general and wide-spread rock alteration if it were not for the intervention of the percolating waters found in all rocks, so far as man has been able to penetrate the earth. These waters convey the heat and other chemically active agencies to the places where they can act. Metamorphosed or altered detritus forms the oldest known rocks of the Lake Superior district; and we know of the original rocks only by the remains of that *débris* now found in them. From the character of that *débris* it appears that here the original rocks were of igneous or volcanic origin; that is, they made up the early-formed crust of the earth, or else were produced by the earth's primitive volcanic activity.

When the muds, sands or shingles have been consolidated, they are found to form rocks that differ not only in the fineness of the material in them, and in their chemical and mineral composition, but also according to the different agencies and conditions to which they have been subjected.

Thus it is that the muds have formed the rocks known as the argillites, shales, most schists, and some gneisses; the sands have formed sandstones, quartzites, some schists, and most gneisses; while the shingle generally finds its expression in the conglomerates.

The term argillite is used to indicate those consolidated muds that were largely composed of clay or argillaceous material; but the argillites are commonly known as slates—a term properly applied to an argillite only when it has been subjected to pressure and chemical action to such an extent that it has the property of splitting indefinitely into thin plates that have no relation to the original structural or sedimentary planes of the rock. This property of being cleaved or split is known as cleavage, and it is by no means confined to that variety of argillite known as slate.

When quartz sands form a sandstone which subsequently has been greatly altered or indurated, so that the rock is composed of a very hard, compact mass of quartz grains, this variety of sandstone is known as quartzite.

The terms schist and gneiss are used to designate all those

altered or metamorphosed detrital deposits whose minerals are arranged in more or less parallel bands, along which the minerals tend to lie flatwise or lengthwise, causing the rock to split into more or less regular plates parallel to these bands. These bands (or foliation of the rock) may or may not be coincident with the original planes or lines along which the detritus was deposited (planes of sedimentation), and in the majority of cases in the Lake Superior region do not so coincide. A striking example of this can be seen in the schist north of Teal Lake, where the planes of deposition run approximately northeast and southwest, while the foliation runs east and west.

The varieties of schist are named according to some one or more of the prominent minerals in them, as hornblende-schist, mica-schist, quartz-schist, chlorite-schist, actinolite-schist, etc., for the schists that contain the minerals hornblende, mica, quartz, chlorite, actinolite, etc.

The altered muds, sands or shingles may be found continuous over large areas; or they may be found, like their modern representatives, to pass gradually or abruptly into one another. Thus it is, that a quartzite is found to pass into quartz-, mica and chlorite-schists; chlorite-schist into argillite-, conglomerate and hornblende-schists, etc.

Eruptive, Volcanic or Igneous Rocks.

To obtain a fair idea of rocks of this character it is necessary to do as we have done with the detrital or fragmental rocks; that is, to observe carefully the recent forms and trace out their structure and various modifications and alterations. If this is done, we see that the eruptive rocks are changed or metamorphosed not less, and often more, than the sedimentary formations.

This may be illustrated by allowing some of the molten iron from our furnaces to run at waste over the ground and into the crevices, so as to be left exposed to the air, frost, wind and snow. It would first solidify, then crack or form joints, as all rocks do, and, owing to the action of the air and rain, it would decompose and alter until finally it would form an earthy iron-ore totally unlike the original iron. Why is this? The answer is that the iron, when it passes from its furnace, is exposed in the outside atmosphere to conditions entirely unlike those in

the furnace, and it must change its state to conform with those changed conditions. So, too, the eruptive rocks, coming in a liquid state from the interior of the earth's furnace, cannot endure unchanged the conditions which exist at or near the earth's surface. They are in an unstable condition, and must be made over into a more stable mineral composition. The agencies that produce that change appear in general to be the same as those which alter the sedimentary formations, namely, percolating waters chemically active, pressure, and heat or cold. The first stage is the change from a liquid or pasty mass into a solid one; later there comes a more or less variable alteration that extends throughout the entire mass, and causes variation in the mineral composition and structure—so much so that at times no trace of its original condition remains, unless it be its form or its relative position to other rocks.

It is in consequence of these changes that rocks which were originally peridotites or olivine rocks are now called serpentines, actinolite-schists, talc-schists, dolomites and *verde-antiques;* and that formerly molten basalts are now designated melaphyrs, diabases, gabbros, diorites, eclogites, amphibolites, hornblende-schists, chlorite-schists, mica-schists, amygdaloids, traps, greenstones, variolites, granites, etc. It may here be said that schists result from the alteration of eruptive rocks, as well as by the change of sedimentary ones. It is alteration that causes rocks that were formerly andesites to be named phonolites, propylites, hornblende-porphyries, porphyrites, diabases, melaphyrs, diorites, granites, schists, etc. In the same way what were once trachytes now form felsites, phonolites, porphyries, granites, gneisses, etc.; while the rhyolites, in their alteration, form rocks called felsites, petrosilex, gneisses, granites, quartz-porphyries, etc. It will be inferred from the above that the alteration of eruptive rocks produces, from forms that were originally distinct, forms that are now known by the same name; while, on the other hand, the varieties due to the various changes of a single rock-species are very numerous.*

* See *Bull. Mus. Comp. Zool.*, 1878, V. pp. 39, 40, 275–287; 1880, VII., 1–164, 183–187, 331–565; *Proc. Bos. Soc. Nat. Hist.*, 1877, XIX., 217–237, 309–316; 1880, XXI., 91–103, 243–274, 288–294; 1883, XXII., 412–432, 485–489; 1884, XXIII., 197–211; *Mem. Mus. Comp. Zool.*, 1884, XI., 208 pp.; *Am. Jour. Sci.*, 1884 (3) XXVIII., 94–104; *Science*, 1883, I., 127-130, 541; 1884, III., 486, 487; IV, 111; *Bull. Minn. Geol. Survey*, No. 2, 1887, 159 pp.; *Report of State Board of Geol. Sur.*, Mich., 1893, 196 pp.

The structure of eruptive rocks differs very much according to their composition, and according to the conditions under which they have cooled, whether slowly or rapidly, as well as according to the conditions to which they have been subjected. That is, a mass slowly cooling will be found to contain much larger mineral forms, known as crystals, than a mass suddenly chilled.

The eruptive rocks, in their relations to their associated country-rocks, will also vary according to the conditions in which they have reached their present position relative to the latter.

If the liquid material (lava) forces its way through a rock, filling the cracks that then existed in it, like putty filling a crack in glass, the solidified lava is known as a dike. It is to be observed, however, that when rocks are but little consolidated, the eruptive or liquid material tends to force itself along the planes of deposition of the sediments, or parallel with the foliation, or else to break irregularly through whatever portion of the rocks offers the least resistance. But when the rocks have become solid then the flow more commonly takes place along cracks or fissures in the rocks, which extend across the country, like the cracks made in a thin sheet of ice. Usually these dikes may be distinguished by their being closely welded on each side to the country-rock, which is often indurated or hardened at the point of contact; by their being compact and fine-grained at the junction with the country-rock, thus showing a rapid cooling, due to their coming in contact with the cold sides of the fissure; also by their being more coarsely crystalline or coarse-grained toward the center than at the margin, because of the greater length of time the interior mass would occupy in cooling; oftentimes by the dikes holding, on both sides, fragments of the country-rock caught up in the passage of the lava. The difference between the sides and the interior of dikes is usually less marked in those rocks which contain over 65 or 70 per cent. of silica than it is in rocks containing a less amount. Oftentimes the lava welling up through a fissure will fail to reach the surface, and usually hardens in a wedge or knob-like form; but at other times it flows out over the surface of the earth, in like manner as water passing through a fissure will flow over the surface of ice.

When lava flows out from long fissures and floods the country, such flows are commonly known as fissure or massive eruptions, especially if the flows were attended with little or no explosive action. If the lava passed through a hole or channel like a "blow hole" in ice, and especially if attended with explosive action, it is commonly called volcanic. The massive fissure or quiet eruptions were more common in the earlier days of this earth; the explosive or volcanic eruptions have been more common in later geological time, or recent times. Since all these are manifestations of the same general cause, we shall use the term volcanic to include all eruptive phenomena.

Lava-flows may generally be distinguished from dikes or intrusive masses of lava by the underside of the flow being welded to the country-rock, by its having baked or indurated the underlying rock, and by its holding fragments thereof; also by its conformity to the original surface of that underlying country-rock. The flow is usually fine-grained or compact at its base, owing to rapid cooling; but a short distance from its base it presents a coarser texture, and usually shows the coarsest structure below the center of the flow, at a point which was the longest in cooling. The upper surface of the flow is commonly wrinkled, cellular and slaggy, if it has not been worn off. The overlying country-rock is laid down upon this surface; it conforms to the inequalities of the underlying lava, and generally contains fragments derived from it. The overlying sedimentary rock is not welded to the underlying lava, nor does one affect the other in any way, unless it be by chemical action.

When any explosive action has taken place, ashes and larger fragments of the disrupted lava are strewn about, which may or may not be subsequently worked over by wind and water. Lava, as soon as it is exposed to the waves, is worn away, like any other rock, and we find its worn detritus deposited by its side, ere the main mass has been cooled.

Chemically Deposited Materials.

Every kind of chemical deposit, even sublimation-deposits, can be considered as resulting from the solution of mineral matter through the agency of the chemically active percolating or surrounding waters, which remove substances from one locality to another, or replace one material by another. While

it is not always true, it is a general rule that ore-deposits are most commonly found in regions of eruptive rocks, either ancient or modern, as in connection with such rocks the percolating waters are apt to be warmest and most highly charged with chemically active solvents.

The history of ore-deposits seems to be the result of a *universal law* which affects all rocks of the globe, and may be considered as the one which governs the entire universe, physical, mental and moral.

For the physical universe this can be best formulated, perhaps, as the *degradation and dissipation of energy; the passage from an unstable condition towards a more stable one; the tendency of all things to harmonize with their environment, that is with their surroundings, or with the conditions to which they are exposed.*

This appears to be the law under which the universe has moved from its beginning, and under which it will continue its course uniformly towards the end; no turning back can occur, and no energy once lost can be restored, except by the same Almighty Power which gave it birth.*

Without entering at all upon the question of the source of those rocks which have come from below the earth's surface and are known as eruptives, and which form a large portion of the so-called metamorphic series, it is sufficient for the present purpose to state again that, when they reach the exterior of the earth, their condition is one not adapted to the circumstances in which they are hereafter to exist. For a time, at least, prior to their eruption, they have been under far different conditions from the atmospheric ones on the earth's surface; of necessity, after their extrusion, there will be a constant tendency on their part to conform to these changed conditions. This is manifested most conspicuously in their loss of heat, and in their passage from a liquid to a solid condition. When solid, these rocks may be said to be in an unstable condition, in respect to their temperature, and also to the chemical combinations formed on solidification. Their chemical arrangements, as manifested in their constituent glass and minerals, are such as to necessitate a transference to a condition in which they will be less acted upon by the agencies to which they are exposed

* *Science*, 1883, I., 541; *Lithological Studies*, 1884, p. 1.

on the earth's surface, and this leads to degradation, dissipation and loss of energy on their part. In other words, the rocks tend to pass from an unstable towards a more stable state. The rapidity of these changes depends not only upon the chemical constitution of the rocks, but also upon the special circumstances in which they are placed. In the basic rocks, or those containing much iron, magnesium, calcium, aluminum, etc., the alterations are comparatively rapid, but in the acidic rocks much slower. If rocks of eruptive origin are studied under the microscope, these alterations can be readily traced from their earliest to their latest stages, and are usually found to be in a ratio to the ages of the rocks or to the particular influences to which they have been respectively subjected.

It is these alterations that have led to the multiplicity of rock-names, and to the confusion of nomenclature; lithologists and geologists, as a rule, formerly supposing that as a rock then was, so it always had been and always would be. For example, the lava-flows of Keweenaw Point, which were once identical with the modern basaltic lavas of Mt. Etna and of Kilauea, are now designated, on account of their alteration and age, as melaphyrs, diabases, diorites, etc.; andesite in its changed guise is designated as propylite, diabase-porphyrite, porphyrite, diorite, etc.; rhyolite as felsite, quartz-porphyry, petrosilex, orthofelsite, etc.; peridotites, or olivine rocks, as serpentine, talc-schist, etc.

The propylite of the Comstock Lode, Nevada, is a striking example. The present writer was the first to call attention to the fact that the fortieth parallel propylites were altered forms of pre-tertiary and tertiary andesites.* The position then taken has been fully confirmed by Dr. G. F. Becker,† and by his colleague, Arnold Hague. Dr. Becker further states that every decomposed rock in the district has been taken for propylite.

The above mentioned changes, or alterations, in rocks of the same composition appear to be largely dependent upon the action of infiltrating waters, and their rapidity seems in a ratio to the temperature of the waters or to the solvents contained therein. These alterations appear to consist, in general, of molecular transferences or chemical reactions in the rock-mass

* *Bull. Mus. Comp. Zool.*, 1879, vol. v., pp. 281, 285, 286.

† "Geology of the Comstock Lode and the Washoe District," 1882, pp. 12–150.

as a whole, and are not confined to special minerals; hence has resulted the failure of theories of mineral pseudomorphism to explain rock-metamorphism, or alteration—the pseudomorphic changes in the rock mass being but the resulting accident of the greater and more general metamorphosis. In the process of alteration the original glass of the rock is broken up, forming various minerals according to its composition, while the original crystallized minerals are changed to a greater or less degree, the resulting minerals being quartz, various ores, anhydrous and hydrous silicates, carbonates, etc. In the course of these changes there is everywhere seen a tendency to localize these secondary products, especially the ores, which results in the removal of material of one kind and the deposition of another in its place, or in the filling of fissures and cavities in the rocks. It is not uncommon to find in rocks minute veins that under the microscope show variation in their filling as they pass through different minerals. What has now been described as taking place in one rock takes place in all, and frequently with various interchanges and reactions between the different associated rocks. If we pass gradually from alterations seen in the rock, and from the minute fissures observed under the microscope, to deposits visible to the unaided eye, to joint- or fissure-planes that traverse large masses of rock, or to cavities, or to any condition of rock structure that will admit the chemical deposition of mineral matter, then, whether we have ore-deposits or not seems to depend upon the activity of the alteration and upon the amount and kind of matter brought together. It is well known that valuable ore-deposits are more apt to occur in regions of eruptive and altered rocks than elsewhere.

From what has been said above of the general alteration of rock-masses and the partial localization of their contained mineral matter by percolating waters, it would appear that this cause gives rise to all chemical deposits, even to sublimations, when the water is vaporized.

Instead of the re-deposition of the mineral matter, taken up by the percolating waters, in the rock or in contiguous cavities, it may be borne far away, appearing in spring-, river-, lake-, and ocean-waters, and in deposits laid down by them; precipitation taking place wherever the proper conditions exist.

Of all theories that have been proposed to account for ore-

deposits, there are few that are not correct for some form of ore-deposits in certain localities. While the practical use of these theories is to aid us in understanding the nature of the deposits, as a guide to their exploitation, the difficulty in such a use lies in the abuse of the theories through their indiscriminate application to all deposits. Our rule ought to be to study every deposit thoroughly, and after the study, not before, apply that theory which best answers to the observed conditions, since all theories ought to be generalizations or expositions of observed facts, with a prophecy for the future.

It is not doubted here that all ore-deposits not of a mechanical or eruptive origin can be attributed to the general alteration and change in rocks that result from the general dissipation and degradation of the potential energy of the constituents of the earth's crust, in the universal passage of matter from an active towards a passive and inert condition.

This general alteration manifests itself in a universal chemical or molecular transference—a transference of material, leading to the segregation or localization of the ores in the places in which they are now found; it manifests itself in the deposition of mineral matter in the veins and cavities of the rocks themselves, in deposits from springs, in bogs, lakes, etc. From this it would follow that all ore-deposits not eruptive are, as regards the earth, superficial phenomena and dependent on its external agencies, although they may be deep enough so far as man is concerned.

Again, few of these ore-deposits would be expected, except in regions in which percolating waters and their attendant metamorphism have been efficient agents;* while the various forms of ore-deposits would be associated with, and grade into, one another.

If we start, as all geologists do, with the belief in an originally hot fluid globe, all rocks must have been derived, primarily, from fluid material, which contained all the elements of the metals or ores. The detrital rocks would naturally partake of the metalliferous character of the rocks from which their material came, and each would develop similar changes; while in

* Whitney, *Contributions to American Geology*, 1880, vol. i. "The Auriferous Gravels of the Sierra Nevada of California," pp. 310, 331, 350, 356.

the chemically and organically formed rocks, in accordance with their special conditions of formation, there can readily be suggested agencies for the precipitation of useful ores throughout their mass during the time of their formation, the precipitated ores being gathered up subsequently in all those rocks by the percolating waters.

In order to estimate the value of deductions regarding the source of the ore in an ore-deposit, based on the analysis of the minerals in the adjacent country-rock, it is necessary to look into the question of the origin of these minerals. If the consolidation of rocks be taken as a reference-point, the minerals they contain naturally fall into three classes:

(1) Those of prior origin—foreign.

(2) Those produced by solidification (crystallization)—indigenous.

(3) Those produced later by alterations in the rock-mass, or by infiltration—alteration, or secondary.*

The first can conveniently be separated into two divisions:

(*a*) The minerals that are characteristic of the rock, whatever may be its locality or age.

(*b*) Those that are accidental, as, for instance, fragments caught up during the passage through or over another rock.

Any rock may contain all three classes, or only one or two, as the case may be. A few minerals may be cited in illustration. Olivine in the peridotites is an indigenous mineral, but in the basalts it is foreign, although generally characteristic of them. Serpentine, when not an infiltration- or veinstone-product, is always a secondary, or alteration-product. Hornblende in the recent andesites is foreign, but in the older andesites, and in almost all the older rocks of every kind, it is either a secondary product or is a more or less altered mineral. The micas, feldspars and quartz occur as foreign, indigenous and secondary products. As a rule, quartz is foreign in the modern rhyolites; but in the older rocks of this type, felsites and quartz-porphyries, it is both foreign and secondary. All hydrous oxides and silicates and all carbonates appear to be alteration-products.

Generally, the different modes of occurrence of these min-

* *Bull. Mus. Comp. Zool.*, 1879, V., 277, 278; *Lithological Studies*, 1884, pp. 25-29.

erals can be readily distinguished from one another under the microscope by the characters of the minerals and their relations to the rock-mass.

Further, it may be pointed out that olivine, except in the more recent rocks, is found, as a rule, to be more or less altered to, or replaced by, serpentine, quartz, iron-ore, carbonates, etc.; augite by hornblende, biotite, chlorite, etc.; and feldspar by quartz, kaolin, micaceous and chloritic minerals, etc.

These changes are so common that it is rare to find in the older rocks original minerals that remain unchanged.

Again, almost every mineral in rocks is found to contain inclusions of other minerals, of glass, liquid and gases, thus vitiating conclusions drawn from chemical analysis of the mineral.

Since ore-deposits are, in general, associated with altered or metamorphosed rocks, and occur in regions in which thermal waters have been active, the country-rock would naturally be more or less changed, and sometimes completely decomposed. In the process of the formation of the ore-deposit it may happen that the ore-material will be entirely removed from the adjacent rock, or in this rock there may be deposited ores that never existed in it before, the ore-material having been brought from a distance by the percolating waters.

From the above it follows that chemical analyses alone, either of the country-rock or of its enclosed minerals, lead to unreliable conclusions as to the source of the ores, as they indicate only what now is in the rocks or minerals, and not what might originally have been there. Hence it is an unphilosophical procedure to build any general theory upon such analyses, unless they are carefully supplemented by thorough and accurate geological study of the region in question.

If, by chemical analysis, any accurate deductions are to be drawn regarding the original source of the ores, it seems necessary that we should select those rocks and minerals that are known to be fresh, unaltered and free from any foreign inclusions that would influence the result. Such materials could only be obtained from recent lava-flows, recently-formed limestones, etc.; for no rock that has been exposed for a considerable length of time to the earth's meteoric agencies can, in the writer's opinion, be said to be in its pristine condition. Most analyses of such rocks fail to include tests for sufficiently minute

quantities of such materials as comprise the more valuable ore-deposits to permit, as yet, any general conclusion to be drawn concerning the source of all our metals. The nearest approach to any such analyses is in the meteorites, which are unaltered, and which in composition and structure are closely allied to certain classes of terrestrial basic rocks. These meteorites are found to contain copper, tin, nickel, cobalt, arsenic, zinc, manganese, chromium and graphite, and it is probable that more careful and searching analyses would reveal in them and in the lavas other and more precious metals.

While it would appear probable that the elements of the useful ores were often originally disseminated through the rocks, particularly the eruptive ones with which they are associated, and were subsequently concentrated by the agency of percolating waters, proofs that this theory is correct are yet wanting—the theory resting mainly upon the observed structure of the ore-deposits, their association and the alteration of the adjacent rocks, or, rather, upon the geological conditions observed, than upon any chemical analyses yet made.

To summarize, it may be said that there is no known rock, unless a partial exception be made for rocks in the form of glass, that will not absorb water to a greater or less extent. All these waters are chemically active, whether hot or cold, pure or impure; but it is undisputed that heat, pressure and substances in solution in these waters greatly increase their chemical activity. It has been pointed out that all rocks are modified or changed through the action of the chemically active waters, leading either to the decomposition of the rocks or to a change in their mineralogical composition, and often to a change in structure. It has further been pointed out that the alteration brought about by chemical waters, with or without heat and pressure, has caused rocks formerly of the same character and composition to take upon themselves very diverse forms. It has also been noticed that rocks of entirely different origin and structure, like sedimentary and eruptive rocks, have been so changed by this action that the resulting forms are indistinguishable from one another except by their geological mode of occurrence.

All these changes in rocks have not proceeded without certain mineral matters being removed from one locality and de-

posited in another. A strong tendency is observed towards a localization of the moved mineral product or towards special aggregation of mineral matters, some of which are economically of no importance, while others form useful ore-deposits.

One of the latest phases of the formation of deposits of value has been the filling in of fissures by the water-deposited quartz and other vein-materials, or, in case no crack nor fissure existed, by the removal of the country-rocks along certain lines and their replacement by vein-matter.

Veins thus formed may contain only quartz or other valueless minerals (gangue), or they may hold valuable metals and ores. It is in veins that the gold and silver north of Ishpeming are worked, the veins at the Ropes gold-mine being in serpentine, while the others are in diorite, granite, felsite and schist.

Potsdam Sandstone.

The rocks of the Azoic system were still subject to the same denudation by frost and rain, and by the beating of the waves after that formation had been completed, that the different members of the formation had been subjected to before the entire system was complete. That is, there were deposited about the Azoic rocks mud, sand and shingle, in like manner as such materials are being deposited on our lake-shores at the present time. This detritus, on consolidation, has formed a series of shales, sandstones and conglomerates, which overlie or abut against the Azoic rocks, and are mainly composed of the *débris* of the latter. These sandstones form in Michigan the base of a new system of rocks which is generally considered to be the equivalent of the Potsdam sandstone; hence the rocks are provisionally said to be Potsdam.

South of Carp river this sandstone can be seen lying against the Azoic quartzites and formed out of their *débris*. The Marquette sandstone quarries are in this Potsdam sandstone, and at the base of the quarries may be found conglomerates made up of the underlying Azoic quartzites, diorites, argillites, schists, etc. A short distance south of Hotel Marquette there can be seen, on the west side of Champion street, leading to South Marquette, some rounded diorite- and schist-knobs that have been polished and grooved by former glacial action. These knobs have been exposed by excavations made for the purpose

of obtaining filling for some adjacent ravines. A matter of special interest is the finding of small veins of the formerly overlying Potsdam sandstone that has filled the cracks in these rocks, which formed bosses or little knobs on the shores of the old Potsdam sea. The Potsdam sandstone is found overlying much of the serpentine of Presque Isle where the basement conglomerate is well exposed, although that conglomerate has subsequently been much altered, apparently, by heat and water. The same sandstone occurs further north on the shore, and on some of the islands that overlie the Azoic granite, which has been decomposed for some distance below the base of the sandstone. This decomposition is seen to extend to the boulders of granite and other rocks in the basement sandstone and conglomerate. These changes unquestionably have occurred since the sandstone was deposited, and the percolating waters are apparently the cause of the decomposition. The sandstone extends, with greater or less continuity, along the shores of Lake Superior, around Keweenaw Bay and Keweenaw Point, where it was first designated by the present writer as the Eastern sandstone. On Partridge Island clayey or sandstone fragments occur abundantly in the Potsdam sandstone itself. The writer has seen similar deposits in the process of forming on the Bay of Fundy, in the vicinity of St. John.

The Potsdam sandstone was evidently formed on the shores of a body of water accessible to ocean-tides, as it shows ripple-marks, sun-cracks, rain-drop impressions and mud-flows, which indicate conditions that are only known to exist in localities where the alternate ebb and flow of the tides occur.

The age of part of the Eastern sandstone of Keweenaw Point has been proven to be at least Lower Silurian (Ordovician) or Cambrian and probably Potsdam, by the work of Mr. W. L. Honnold.* A limestone west of L'Anse in T. 51, R. 35, has been described in Jackson's Report (1849, pp. 399, 452), Foster and Whitney's Report (Part I., 1850, pp. 117–119), and in Rominger's Report (1873, I., part 3, pp. 69–71), and the limestone has been considered, from its fossils, to be of the Trenton or some adjacent lower strata. It was inferred by Jackson that the limestone underlies the sandstone, but by other observers that it overlies it, although no direct contact was seen.

* *Am. Jour. Sci*, 1891 (3) xlii., 170, 171, 417–419.

Excavations made by Mr. Honnold's party and reported by him have exposed the contact of the two formations, and show that the two form a synclinal or oblong basin-shaped fold, with the limestone overlying and in direct contact with the sandstone. The existence of this fold in the sandstone, as well as in the limestone, removes the difficulty previous observers have had in reconciling the obviously tilted limestone with the supposed horizontal sandstone, and proves that the Eastern sandstone exposed here is of Lower Silurian (Ordovician) or Cambrian age, and older than this limestone.

At the point of contact of the two formations, exposed by excavation, the sandstone and limestone appear to be conformable, and they are seen constantly to agree in dip and strike. The contact between the two formations is abrupt, without any beds of passage, although the upper layers of the sandstone contain considerable carbonate of lime and magnesia, and the lower layers of the limestone much silica.

These observations are considered to be confirmatory of the commonly received view of the Potsdam age of the Eastern sandstone; while the contorted state of the sandstone which extends at least a mile and a half west from the limestone locality may have weight in deciding the relative age of the Eastern sandstone and the copper-bearing rocks.

During the time of the deposition of the Eastern sandstone (Cambrian), or previous to it (Algonkian), volcanic activity again commenced, and the central portion of Keweenaw Point is found to be composed of alternating lava-flows, sandstones and conglomerates, deposited upon the tide-washed sinking shores of the sea. The intermittent volcanic activity ceased for a while after the main range of Keweenaw Point was produced, leaving time for the formation of a broad belt of sandstone and conglomerate; but it again recommenced, making the basaltic rocks exposed along the northern side of Keweenaw Point at Eagle Harbor, Agate Harbor, Copper Harbor and elsewhere.

In connection with these lava-flows from fissure eruptions, which were of a basaltic character, there was also extravasated much volcanic material of a trachytic and rhyolitic nature, the *débris* of which makes up the chief portion of the interbedded sandstones and conglomerates. These occur in the form of intrusive dikes, bosses, etc.

The basaltic rocks forming the southeastern range of Keweenaw Point, known as the Bohemian Mountains, were considered by Foster and Whitney to be intrusive masses erupted subsequently to the formation of the main deposits in this region. Irving, however, considered them to be ordinary flows, like the rest of the lava-flows of Keweenaw Point. He does not advance any special proof of this proposition, while the phenomena presented by him, as well as by Foster and Whitney, appear to be more in accordance with the view that they are later eruptive masses, as held by the last-named observers. However, the question is yet an open one. The basaltic lava-flows are known to be such, as pointed out by Foster and Whitney, and later by Marvine and Wadsworth, by their baking, or indurating, or hardening the underlying rock; by their sending dikes and tongues down into the rocks; by their having caught up fragments of the underlying rock; by their crystalline structure being best developed below the center of the flow; by their having no effect upon the overlying conglomerate, while the *débris* of the lava is to be found in the base of the conglomerate; by the overlying conglomerate and sandstone filling cracks or forming clasolites* in the underlying lava-flow, etc. The thinner basaltic lava-flows were cooled quickly, so that they contained much glass, which was readily decomposed by the percolating waters. In their altered condition they now form rocks known to geologists as melaphyrs, but which from their present structure are locally called amygdaloids.

The thicker flows formed heavy beds, which cooled more slowly, became more crystalline and were less easily affected by the percolating waters than the thinner flows. These heavy flows, owing to their alteration, now form rocks which by geologists are named diabases and gabbros, but locally called traps and greenstones. All of these melaphyrs, diabases and gabbros were once lavas of the same chemical constitution, differing only in structure and in those differences of crystallization and mineral constitution that result from slow or rapid cooling.

* See *Report of State Board of Geol. Sur.*, Michigan, 1893, pp. 129, 130, 147, where the present writer defines a clasolite as a sedimentary deposit of mineral matter of indefinite length and depth, and comparatively small thickness, differing in character from, and posterior in formation to, the rocks which inclose it, *i.e.*, the so called sandstone dikes of Diller and others.

These flows must have taken place over the gently sloping tide-washed shore of the sea, which shore was gradually, or it may be at times abruptly, sinking, so that the flows and their detrital deposits accumulated at about the same rate as the shore sank, making the shore-line approximately constant. This must have been the case, or the lava-flows and conglomerates would have been more irregular and less constant; they would have covered a more limited area, and would soon have been built up beyond the reach of the waves.

Owing to the natural irregularities of a lava-flow, and the resulting inequality of the sedimentary deposits, it is to be expected that some inequalities in thickness of both formations should exist, and that sometimes one or the other should be wanting. For instance, if a portion of the lava was raised above the sea, that portion would not be covered by either sandstone or conglomerate, but only by its own decomposition products, if even they were not carried away by the rain. Hence it happens that the interval between two flows is marked at one point by a conglomerate, but elsewhere only by a thin seam; or, as is often the case, the two flows are interfused.

It also frequently happened that a comparatively short time existed between two flows. In such a case little or no conglomerate could form between them, and as the latter flow fused again the top of the earlier flow, the two became united into one mass, so that it is difficult to ascertain where one begins and the other ends.

The Copper-Bearing or Keweenawan Series.

The relation of the lava-flows with their interbedded conglomerates to the Eastern sandstone is a matter of great scientific and economic interest, and has given rise to much discussion, which is likely to continue for many years to come, until the whole truth shall be known. In the report of Foster and Whitney, the Eastern sandstone was considered to have been once continuous with the Western sandstone* of Keweenaw

* These designations for these sandstones were first used by the present writer and later adopted by Irving. *Notes on the Geology of the Iron and Copper Districts of Lake Superior*, 1880, pp. 93, 107, 108, 110, 116, 121, 122; *Proc. Am. Assoc. Adv. Sci.*, 1880, xxix., p. 429; *Proc. Bost. Soc. Nat. Hist.*, 1880, xxi., pp. 92, 94, 103. "Correlation Papers. Cambrian." *Bull. U. S. Geol. Sur.*, No. 81, 1891, pp. 252, 335.

Point, but it was thought that the two parts had been separated by a fracture or fault-plane that extended along the entire southern side of Keweenaw Point. This fault allowed the sandstone on the east to remain horizontal, while the lava-flows on the west were tilted up at their present angle, and the overlying western sandstone was subsequently worn away. The sandstone east of the fault-line was said to lie horizontally, or to dip to the southeast.

Later, attention was called to certain observations made along the line of the fault, especially at the Douglass Houghton Falls. These observations caused the lava-flows and their interbedded sedimentary rocks to be considered as an older geological formation than the Eastern sandstone. It was said that the lavas formed an old sea-shore bluff on the Potsdam sea, and that the sandstone was laid down horizontally, abutted against the cliffs, and was formed from their water-worn *débris*. Various other opinions have also been advanced, reference to which our limited space does not permit.

In 1879 the present writer made an examination of the formations at the points at which, on the Douglass Houghton and Hungarian rivers, the Eastern sandstone comes in contact with the lavas, or, as they are now commonly called, the copper-bearing rocks or Keweenawan series. He found that the sandstone, instead of lying horizontally, dipped gradually or irregularly toward the northwest; and that, instead of abutting against the copper-bearing rocks, it was overlain by the latter, and the two were interbedded. Later, the correctness of these observations was denied by Irving, who upheld the view of the greater age of the copper-bearing rocks, but moved the supposed sea-shore cliff from its former supposed locality and placed it elsewhere. Subsequently the question was taken up by Irving and Chamberlin in defence of the view that the copper-bearing rocks are older than the Eastern sandstone. But upon studying their published observations, together with their sections given in this later work, it will be seen that their preceding observations are discredited, and that they fully sustain the correctness of the present writer's statement that the sandstone dips under the copper-bearing rocks, instead of being separated from them by a vertical fault or an old sea-bluff. The result is that in the main point at issue the present writer was shown

to be correct, *i.e.*, that the sandstone does underlie the copper-bearing rocks, and then the main question was transferred to one of interpretation of the observations. Do the lavas overlie the Eastern sandstone on account of their having flowed over it in the form of a molten lava, as the writer then claimed, or had the lavas been thrust up over the sandstones through the motion of an older solid mass along an oblique fault-plane, as last held by Irving and Chamberlin?

During the summer of 1889 an examination of this question was made by the Michigan Geological Survey, then under the writer's direction, and the rocks were uncovered along the line of contact of the sandstone and lavas for the purpose of ascertaining their exact relations. The result has been to prove to the present writer, beyond any reasonable doubt, that on the Douglass Houghton and Hungarian rivers, as well as on Sec. 20, T. 56, R. 32 N., the sandstone does dip gently toward the lavas, and finally passes under them with an increasing dip; that the junction is not a fault-junction but that of a lava-flow upon an underlying soft sand and mud. The lowest bed of melaphyr was found on the Douglass Houghton river to be overlain by sandstone, as described by the writer in 1880, although this fact had been denied by the before-mentioned authors. The copper-bearing rocks on the north side of Douglass Houghton river are seen to overlie the Eastern sandstone for about 150 feet. Along the line of contact to the north on the St. Louis location and in Wall ravine, as well as south of Portage Lake, proof of distinct faulting could be obtained, which sustains the fault claimed by Foster and Whitney, Irving and others. Along the Douglass Houghton contact, no signs of any fault, or but few signs, were found; but in the copper-bearing series, just below the falls, several fractures, with evidences of the motions of sides, were observed, that would indicate a faulting here.

The fault-plane, wherever observed along the line of contact, showed that the hanging-wall was on the side next the copper-bearing rocks. It is well known by all miners and geologists that in all normal faults, *i.e.*, the commonly occurring faults, the rock on the hanging side of the fault has slipped down relatively to the rock forming the foot-wall. A reversed fault is one in which the hanging-wall side has been pushed up

on the foot-wall side of the fault. The reversed faults are generally considered to be rare; so much so that some geologists deny that they ever occur, although the present writer has seen them associated with normal faults in the Cheever ore-bed at Port Henry, New York. This rare mode of faulting is the one assumed by Irving, who was obliged also to assume that the uplift of the hanging-wall was accompanied by a thrust to the eastward: a view that some observations by the Michigan Geological Survey, especially by Professor A. E. Seaman, would sustain; but these, the present writer thinks, need more careful examination and further confirmation before they can be accepted as conclusive, since all the observed facts can be explained by the repeated movements that usually occur along the sides of a fault.

The present writer holds that the sandstone underlies the copper-bearing rocks, and that the first lava of that series flowed over the Eastern sandstone, which is older than the copper-bearing or Keweenawan series. Subsequently a fault-line or fissure was formed, running near what is now the point of contact of the sandstone and lavas, sometimes exactly at that point, sometimes on the lava side, and probably sometimes on the sandstone side of it. Along this fissure it is probable that a normal fault occurred, along which, by the slipping down of the hanging or wedge-shaped side, the sandstones and their interbedded lavas were more or less bent downwards or contorted, as they are now found to be. This normal faulting accounts for the fact that sometimes the lava and sometimes its associated conglomerate is brought in contact with the Eastern sandstone along the fault-line. It is to be remembered that in almost all faults there is more or less rubbing back and forth, or up-and-down motion, although the final result of these varied motions is the production of a reversed or normal fault, according to the direction in which the greatest amount of motion took place.

This view would explain most of the difficulties that geologists have had in understanding this series, especially if it should be shown that the lava-flows came from the main lake-side instead of from the Keweenaw Bay side, as that would only require the cut-off remnants of the edges of the lava-flows to be removed by denudation on the Keweenaw Bay side.

Should the reversed fault be proved to be the true one, then the view of Foster and Whitney concerning the relations of the copper-bearing rocks would appear to be more correct than that of Irving, while a normal fault would be consistent with the theory of Houghton, Jackson and Marcou, that the copper-bearing rocks are of Triassic age.

Veins and Copper-Deposits.

Besides the fault before mentioned, numerous fissures cross Keweenaw Point instead of running longitudinally with it, and more or less faulting occurs along these fissure-lines. Portage Lake lies in a trough excavated along one of these fissures, while many of the others are filled with vein-matter, which has been mined to a greater or less extent. These fractures and fissures, with faulting across Keweenaw Point, probably were developed subsequently to the formation of the longitudinal fault or faults, if more than one such fault shall later be proved to exist. Should such be the case, it would account for part of the assumed thickness of the beds.

The before-mentioned fissures seem to have been formed by powerful movements of different parts of the rocks that caused the cross-fracture and dislocation of the latter. The movements were repeated from time to time, causing a rubbing, grinding, breaking and polishing of the parts adjacent to the fissures. After the main fissures had been formed, the subsequent movements would not cause any very extensive secondary breaking of the compact and heavy beds of diabase and conglomerate, but in the soft and scoriaceous melaphyrs the fracturing would be much greater, so that the parts adjacent to the fissures would be much broken. During the time of the fracturing, and subsequently, these fissures served as channels for the chemically active waters, which also percolated through the adjacent rocks. In the scoriaceous and easily decomposable melaphyrs the veins were widened by the decomposition of the adjacent rock, but in the coarsely crystalline and heavy diabases, as a rule, the same effects were produced either not at all or only to a limited extent. The sandstones and conglomerates, being composed principally of trachytic and rhyolitic material, are much less affected by percolating waters than the basaltic rocks; hence the fissures are not generally widened

in them, especially if they are in thick beds. At the time when the percolating waters were acting on the rocks adjacent to the fissures, they were also working in the rocks everywhere upon Keweenaw Point.

In many localities the evidence is strong that the percolating waters were hot, while in others, as remarked by Marvine, no sign exists that they were above the temperature of the waters of the present day. These waters percolated with more or less readiness through the rocks, causing a greater or less alteration and decay, while the substances they took up were deposited in any fissures, cells or other open spaces that existed in the rock, or else portions of the rock were dissolved out and their places were refilled. This is strikingly seen in the conglomerates, like that of the Calumet and Hecla, in which pebbles of the easily-decomposable melaphyr have been partly or entirely removed and their places filled with copper or some other minerals.

Besides copper, the deposited minerals are mainly quartz, calcite, epidote, laumontite, prehnite, delessite, chlorite, datolite, analcite, orthoclase, apophyllite, etc. All, or nearly all, of the materials that fill the crevices, cells and other places in the lavas and conglomerates apparently were derived from the decomposition of the lavas themselves, and the course of the waters carrying these materials in solution seems to have been downward. The fissures that form the veins were filled at the same time and by the same agencies as those that acted upon the rocks, and the materials in them likewise appear to have been obtained from the adjacent rocks themselves. In the narrower portions the veins are often filled with the vein-matter proper, but in the wider portions the veins are often composed of broken-up masses of melaphyr, etc., cemented by vein-matter.

In the veins the copper is found intimately mixed with the gangue, or in sheets or irregular masses. In sheet-form the copper extends downward or has its sides approximately parallel with the vein. Often the sheet is divided, and holds between its parts some of the gangue or melaphyr. It is also not uncommon to find, entirely enclosed in the copper, masses of melaphyr, quartz, calcite or other vein-materials. The melaphyrs themselves are often impregnated with copper adjacent

to the veins. Good illustrations of the veins can be seen at the Phœnix, Cliff, Central, Copper Falls and other mines in Keweenaw county.

In the vicinity of Hancock, Houghton and Opechee some of the old lavas are mined. As stated before, these old lava-flows, which now form melaphyrs (amygdaloids), have been greatly acted upon by water, and have had deposited in their mass different minerals associated with more or less copper. The copper is generally deposited in an irregular manner in the melaphyr, forming strings, globules, irregular masses, etc. These deposits are not in the form of veins, but are impregnations of old lava-flows, and hence are in the form of beds. As an example of mines worked upon these old lava-flows there may be cited the Quincy, Franklin, Osceola, Atlantic, Huron, and the Copper Falls in part. The Copper Falls has been worked in part on an old lava-flow of a very scoriaceous character that formed originally a black, rough, cellular lava-sheet covered with clinkers, similar to many well-known modern lava-flows. At the time of the flow, or after it, the interstices were filled with detrital mud, while the various parts of the flow appear to be connected, and do not form true water-worn pebbles. The writer has collected at Copper Falls portions of the rock that show the hardened exterior crust and the cellular interior, as they occur in small masses and bombs of modern lavas, while he has found preserved intact the original ropy, stringy, twisted surface of the lava. The Copper Falls bed, above described, is locally called the ash-bed, but it is a melaphyr or a true lava-flow, and not a bed of volcanic ashes. The Atlantic mine appears to be worked upon the same or a similar formation.

Besides the veins and lava-flows, the conglomerates have also been found in places to have had their interstices filled in with copper and other minerals. In them the old cementing mud, and oftentimes the pebbles of melaphyr, have been removed by percolating waters, and their places filled with copper, which penetrates even the minute cracks in the hard rhyolite (quartz-porphyry) pebbles. These old sea-beach conglomerates are now worked in the Calumet and Hecla, the Tamarack, the Allouez and other mines. There are thus mined in this region three distinct classes of deposits:

Copper-Deposits of Keweenaw Point.

1. Vein-deposits (fissure-veins).
2. Flow-deposits (melaphyrs or amygdaloids).
3. Bed-deposits (conglomerates).

The conglomerates are known to be old sea-beach deposits, like those that are forming along the lake or ocean at the present time. This is proved by the rounded and water-worn character of their pebbles and grains; by the observed water-action on the surface of the underlying lava-flow; and by the fact that at their base the conglomerates contain considerable basaltic mud and pebbles derived from the underlying lava, both of which diminish in amount or are entirely wanting as the distance from the underlying trap increases.

That the copper was deposited from water, with or without electro-chemical action, is shown by the fact of its being found inclosed entirely in minerals known to be formed by water only; also by its inclosing such minerals; by its being found in disconnected or isolated masses in the lavas and elsewhere, and by its greater abundance where there are to be seen the most signs of water-action. Had the copper been deposited by igneous agencies subsequently to the formation of the melaphyr and conglomerates it would have had a channel or line of passage, and would have been continuous along that line, so that all the different masses of copper would have been connected together downward, unless separated by fractures or faults.

The copper seems to have needed for its deposition the open spaces of veins and fissures, and rocks that were porous and cellular, or those whose parts were easily removed by the percolating waters, like melaphyrs or the cementing mud of the conglomerates. In truth, the copper seems to have been deposited wherever there were found any places in which to leave it.

From the fact that the copper is generally found most abundantly under the heavy lava-flows, and associated with minerals evidently the product of the decomposed lavas, it appears probable that the copper was once finely disseminated through the lavas, and has since been concentrated by waters percolating through them. This view is advocated by Müller, Bauermann,

Marvine and myself, while a similar view has been advanced by S. F. Emmons to account for the origin and concentration of the Leadville ore-deposits.*

Had the copper been derived from the sandstones, then one would suppose that under them should be found the greatest supply of copper; but such is not the case. That the course of water depositing the copper was generally downward is indicated by the finding of spikes of copper and calcite that extend from one bed down into others, with the small end downward, like an icicle; by the fact that when the copper is not uniformly distributed throughout the bed or flow that is mined it is often, although not always, more abundant in its upper portion; and by the fact that the largest masses of copper have usually been found in the upper portions of the veins.

That the copper was deposited after the copper-bearing series was complete is shown by the fact that it is found in fissures extending across the beds that could only have been produced after the beds were in place; by the fact that the copper was deposited subsequently to the jointing of the lavas, owing to its now being found wrapped around the pieces formed by jointing; and by the extension of the copper from one flow down into another as a continuous mass.

The means by which the copper was concentrated and deposited as native copper, instead of occurring in the form of the usual copper ores, is an interesting and as yet unsolved problem that awaits the attention of the chemist who is willing to give his time and thought to the subject, although Pumpelly advocates the idea that the principal agent is the oxide of iron. In this he has much to sustain him, and his view is generally adopted.†

The structure of Keweenaw Point may, then, be summarized as follows: A deposit of sandstone overlain by lava-flows mingled with more or less of interbedded conglomerates, and finally overlain by sandstones. Subsequently these beds suffered longitudinal and cross-fracturing and faulting. Later all

* Müller, *Verhl. d. Natur. Gesell.*, Basel, 1854–57, pp. 411–438; Bauerman, *Quart. Jour. Geol. Soc.*, 1866, XXII., 448–463; Wadsworth, *Notes on the Geology of the Iron and Copper Districts*, 1880, p. 126; Emmons, *Geology and Mining Industry of Leadville*, 1886, pp. 378, 379.

† *Geol. of Mich.*, 1873, I., part II., p. 44.

were acted upon by percolating waters, both hot and cold, by which the rocks were altered to a greater or less extent, and the copper was concentrated and stored up in the conglomerates, lavas and veins in which it is now found.

The above account gives in brief a general idea of the geology of the region touched upon here, as the writer interprets the facts observed. He is, however, aware that different interpretations of the same facts are made by others. He has therefore called special attention to the important differences of interpretation. In the copper-bearing rocks, the question of their relation to the Eastern sandstone is one of great economic interest in these days of diamond drills and deep shafts. One can readily see this when one considers that it involves the question, Do the copper-bearing rocks extend out under the Eastern sandstone or not? If they do, their exploration becomes merely a question of how great thickness of sandstone must be bored through. If they do not, then the question ought to be settled by the geologist, if possible, in order to save waste of money in unnecessary exploration on the part of those who are interested in mining.

The Michigan College of Mines.

BY DR. M. E. WADSWORTH,* HOUGHTON, MICH.

(Lake Superior Meeting, July, 1897.)

THE Michigan State College of Mines was established ten years ago last September as the fourth and last of the institutions of Michigan which are devoted to higher education. From the moment of its inception its single object has been to send out men who are qualified to take active part in the development of the mineral wealth of our State and nation, and any study which is essential to this end will not be found wanting in its courses of instruction.

The institution has been singularly happy in its location. It is an axiom of modern education that any school which is to obtain the greatest return for the money and energy spent in

* President of the Michigan College of Mines.

establishing it must be situated in a region which shall, from its very nature, serve, free of all expense, as a part of the real equipment of that school. This can be so only when the district presents, for the daily observation of the student, the most exhaustive and "up-to-date" practical applications of the subjects which are taught in the school—a district in which the student is inspired with a strong appreciation of, and interest in, his future work by being brought into frequent contact with able men whose lives have been devoted to the same profession. The location must be where the body of the people regard the institution as a co-worker with them in the main business of their daily life, and naturally take far more interest in it than in a school whose object and officials have little in common with them.

The student thus lives in an atmosphere which harmonizes with his work, and his attention and energies are not distracted and wasted by making digressions into matters which have no bearing on his real future.

On this account medicine, law, theology and cognate subjects are best taught in large cities, whose hospitals, libraries, courts, churches, societies and meetings of congresses devoted to such subjects, all furnish material indispensable for the proper teaching of these branches of knowledge. Mechanical and electrical engineering can be most advantageously studied in a locality containing machine-works, manufactures, mills and lighting- and power-plants; and civil engineering can be most thoroughly acquired in a school which has in its immediate vicinity numerous examples of modern constructions in the form of bridges, railways and their appurtenances, large buildings, etc., and the establishments which design and produce the material for them. Similarly, mining engineering can be most effectively taught only in a district containing numerous mines which are worked on the largest scale, and in which full sway is given to scientific and economical considerations.

From this point of view the Michigan College of Mines has been more fortunate in its location than any other school of mines in America. Within a radius of eleven miles from its doors are situated some of the largest, deepest and most successful mines in the world. The most powerful and stupendous machinery ever

employed in mining is here in constant use, and always open to the inspection of the student. The aggregate horse-power of the engines used by only two of these mining companies exceeds the grand total of all the engines used in the gold- and silver-mines of the United States in the halcyon days of 1889, or far more than double all those employed in the grandest spectacle this earth has ever seen—the Columbian World's Exposition of 1893.

The mines of the Marquette, Menominee and Gogebic iron ranges are within a few hours' ride of the college, and furnish a most efficient means of illustrating a large part of its teachings. The value of the annual output of the various mines and other mineral industries in the State, and mainly in the Upper Peninsula, reaches the enormous total of $70,000,000, and places Michigan among the States of the Union second only to Pennsylvania in the value of its mineral products.

The further development of this vast industry requires men who are thoroughly trained for this special work, and to meet this requirement the Michigan College of Mines was established. It was fortunate for the college that its founders realized at the start that to fulfill its mission it must not, as is so frequently and unwisely done, waste its energies in the vain attempt to teach branches foreign to the object of the institution, and which are already well taught by other colleges maintained by the State. In consequence; they determined that this school should confine its attention wholly to *mining and the subjects relating thereto.* Hence the course is designed to teach the student to conduct explorations in the forest and field; to distinguish the useful minerals and rocks; to understand the geological principles that govern the formation and association of useful mineral products, and to determine approximately their values; to study ores, building-stones, limes, cements, mortars, coal, salt, gypsum, petroleum, natural gas, clays, fertilizers, gems, and useful vegetable products; to survey, map and lay out the ground, the railroads, tramways and towns; to select or design hoisting-, transportation-, power- and light-plants; to design the mills, furnaces, docks, dams, bridges, shaft- and rock-houses and other structures; to determine in each case which is the most suitable method for opening and conducting a quarry or mine, and of timbering, ventilating and

draining it; to assay, concentrate and smelt ores; to investigate the strength and other properties of engineering materials, that designs may be intelligently worked out; to make working-drawings to illustrate fully these designs; to understand the most economical methods of generating and using steam; to study in detail engines, pumps, boilers, and other machinery, and the methods of operating, testing and repairing them; to master the principles of electricity and its generation, storage, transmission and use as an illuminant and source of motive power; to study hydraulics, and its various applications in civil affairs and hydraulic mining; to understand mine-management and accounts—in short, to train men to be of real use in any line of work connected with the winning and reduction of mineral products.

Such work naturally arranges itself along various clearly-defined lines, in each of which the training may proceed to almost any length. Hence the branches of study which here most naturally resolve themselves into specialties are drafting; chemistry; assaying; metallurgy; mechanical, civil, electrical and mining engineering; ore-dressing; mineralogy; petrography; and geology. In consequence, the men educated at the Michigan College of Mines are now engaged as surveyors; mining, civil, electrical and mechanical engineers; woodsmen; explorers; railroad-men; chemists; assayers; mill-men; quarrymen; manufacturers; stock-raisers; farmers; teachers, etc. It therefore appears that the college has fulfilled its mission and educated its students to be useful and efficient men, who are daily adding to the world's material wealth and to its stock of knowledge.

The range of subjects bearing upon the mineral industry is extremely wide; and this, coupled with the fact that all men are endowed with a natural aptitude for some lines of work while wholly unfitted for others, and the further fact that circumstances beyond control frequently force men into particular occupations and deprive them of the opportunity to prosecute a full course of training—all conspire to demonstrate the correctness of the modern educational view that a rigid, set course of instruction in higher education is now out of date, and that the student must have some liberty in selecting the studies which are to enable him to cope with his life-work. This fact has been

long recognized here, and the Michigan College of Mines has squarely met the issue by adopting an elective system so designed that, while the student is allowed to choose the main lines of his work, he is compelled by the proper sequence of studies to take up such subjects as are essential to a broad and thorough engineering education. In this respect it stands alone amongst American engineering colleges.

Further, in the effort to save the valuable time of the young men, the school-work is continued during forty-five weeks of the year, instead of from thirty to thirty-six, as in most institutions. This enables the student who wishes to do so to accomplish as much in three years as he would ordinarily do elsewhere in four, and graduate one year earlier.

The instruction given at the Michigan College of Mines is intended to be strictly professional and practical, and the college considers that to give a general educational training is as little in keeping with its legitimate functions as would be the case in schools devoted to law, medicine and theology. This kind of training is already fully provided for by other institutions maintained by the State, and to attempt it here would cause a serious deterioration in the quality of the engineering instruction and add only another source of expense to the State. It is clear that the Michigan College of Mines in no way encroaches on the work which legitimately belongs to any other of our State institutions of learning; it rather assists them by offering an education which not one of them can possibly give in the way it is given here. The mineral industries adjacent to the college are amongst the most efficient portions of its equipment. Without a similar advantage no college can properly teach mining, and any attempt to do so will result in an inferior engineering training, no matter if millions of dollars be spent in equipping the college itself.

The Pennsylvania Railway Company wisely saw this principle when it established its school at Altoona. It wanted railroad men, and it very properly trains them at a place where all matters pertaining to railways are directly under the eye of the student.

The Michigan College of Mines forces its students to do thorough work, and also to acquire some practical knowledge of the subjects they are studying. The day has passed when

engineers could be made through an equipment of lecture-rooms, teachers and books. This college has therefore ever been mindful of the wise observation of Seneca: "Long is the road to knowledge through precepts only; short and effective through practical examples;" hence its students are taught not only to hear, but to see, and to use not only their minds, but their hands as well. In this fact, and in it only, can be found the reason why its graduates have been signally successful in practice, and that they have rarely been diverted into a line of work different from that which they pursued at the college.

The Michigan College of Mines was established by an act of the legislature, approved May 1, 1885, and was opened for the reception of students September 15, 1886. Its inception, establishment, and, to a great extent, its support, have been due to the foresight, energy and executive ability of Hon. Jay A. Hubbell, of Houghton. He has spared no labor in endeavoring to accomplish everything he deemed essential to the success and prosperity of the institution.

Other active and energetic members of the Board of Control have been Messrs. A. Kidder and J. M. Longyear of Marquette; J. N. Wright and P. C. F. West of Calumet; T. B. Dunstan and J. R. Cooper of Hancock; T. L. Chadbourne, G. Pope and J. Senter of Houghton.

In 1886 Albert Williams, Jr., a graduate of the College of New Jersey, was elected as principal and had charge of the school until he resigned, in the summer of 1887. M. E. Wadsworth, a graduate of Bowdoin and post-graduate of Harvard, was chosen as president, and still remains as the chief executive officer.

Of members of the faculty who have moulded their departments and left their marks thereon there may be mentioned Prof. R. M. Edwards in civil and mining engineering; Dr. H. F. Keller in chemistry; Dr. Edgar Kidwell in mechanical and electrical engineering; Dr. G. A. Koenig in chemistry and metallurgy; Prof. F. W. McNair in mathematics and physics; Prof. A. E. Seaman in mineralogy and geology; Prof. F. W. Sperr in civil and mining engineering; Prof. A. E. Haynes in mathematics; and Mrs. F. H. Scott in the library.

The first classes were taught in rooms located on the top floor and in the basement of the "fire-engine house" of the village of Houghton.

Four additional rooms in the Odd Fellows' building were secured in September, 1887, but the continued growth of the school made it necessary to vacate the latter quarters during the summer of 1888 and to replace them by others obtained in the Roller Rink building, now the Armory Opera House.

In May, 1889, the school was moved into the building now known as Science Hall. This hall was erected by the State on land donated by Judge Hubbell, but it was even then well known that the building was too small for the necessary work of the school. In 1890 ore-dressing works were constructed, and there was added, in 1892, a small structure containing a furnace for roasting ores. As the school grew faster than its most sanguine friends had any reason to hope, further buildings became necessary, and during 1894–95 the State erected another large one to accommodate the departments of drawing and mechanical and electrical engineering, and the offices and lecture-rooms of the department of civil and mining engineering. It has also made provision for the erection of a commodious assay-laboratory.

The equipment of the institution has also been increased, so far as its funds have permitted, and the total college property, inventoried last fall, amounted to $231,895.97.

In 1886 a course of instruction of two years' duration was announced, but was not prepared; indeed, the educational side of the school was not regularly organized until the school year 1887–88. Since then the curriculum has been steadily broadened and perfected in detail. In 1889 a three years' course was adopted, and in 1893 this was changed to four years. The full elective system went into successful operation in 1895, and has proved entirely satisfactory to instructors and students alike.

At the time the present president took charge of the educational side of the Michigan College of Mines there was no educational policy developed, and no course of instruction had been laid out, beyond the statement that the course should be two years in length. At that time there were only two distinct State schools of mines in operation, besides the Michigan College of Mines, neither of which confined itself to mining-engineering subjects. The Colorado State School of Mines was established in 1874, and in 1887–88 it had in all its courses 45 students. The Missouri School of Mines, with its preparatory

and all its other departments, had in that year 46 students. There were numerous departments of mines in connection with various State and other universities and colleges; but the numbers of students of mining in all of these, except Columbia and Lehigh, were much less than those given above for Colorado and Missouri. Columbia had, in 1887–88, 51 students in mining, and Lehigh 36.

The way was dark and forbidding, and the remains of past failures could be seen everywhere. In truth, no State school of mines that was ever organized in the United States could by any exertion of imagination be called successful up to that time. Success could not be hoped for except through the locality of the school and a rigid adherence to principles to be laid down then and steadily carried out. Locality alone could not do this; there was also needed an organization that should attempt to save the good and avoid the evils of other institutions.

The main principles then formulated for the reorganization of the school were as follows:

1. To keep the school a special one for giving instruction in all branches relating to the development of the mineral wealth of the State and nation.

2. To give the very best, most practical and highest education in this field that it was possible to reach with the means at command.

3. That the instruction should always be by laboratory and field methods, or a true union of theoretical and practical instruction.

4. That the school should, in its earlier days, put every dollar obtainable into equipment and collections for teaching purposes, and spend nothing for show, until the departments themselves were in a condition to do their proper and designated work.

5. That special opportunities should be given to men of mature or advanced age who had been engaged in practical work, and who wished to obtain an education to aid them in their subsequent labors.

6. That the catalogues and other publications should state, so far as can be known in advance, the exact truth about the school and its instruction.

7. That no one should be appointed an officer in the school,

or be retained therein, for any other reason than his fitness for the place and his capabilities for doing his work.

8. That students in the school must work or leave; and that no distinction should be made among them on account of their worldly wealth or honor, or birthplace—the only criterion should be their ability to perform their duties. Also that quality and thoroughness of work should be the aim, and not mere numbers of students.

9. That the institution should be managed on business principles, and the professors be allowed to conduct their departments according to their own individualities. They should be held responsible for producing the required results, but not for their methods of bringing about these results. They should have entire choice of, and control over, their subordinates, who are to be solely responsible to them. Further, that the professors should receive from the president every aid and assistance possible in developing their departments, and that they should be supported in enforcing proper discipline.

10. That the school should be conducted for the benefit of the students and of the State and nation, and not for the particular advantage of the town in which it happened to be located, or of any special clique. This would be done in the belief that in serving the State best the local town will in the end be more benefited than by any other policy.

So far as it has rested with the president, these principles have been put into execution. The result has been eminently satisfactory, since no college in America has had in its course in mining engineering such a phenomenal rise and development as this Michigan one, whether viewed from the standpoint of attendance of students in mining engineering, or in the thorough and practical nature of the education given, or in the standing the college has won at home and abroad. Since 1890 the Michigan College of Mines has been recognized as one of the leading institutions of its kind in the world, and it will ever strive to retain this position by hard work and true merit, if those for whose benefit it was established will do their duty toward it.

All credit is due to the professors and their assistants who, working under these principles, have so nobly aided in building up this strong and flourishing college; to the able Board

of Control who have assisted and sustained them; and to the noble State that has furnished the means to build an institution of which the State and nation may most justly be proud.

The application of the preceding principles has had such an effect upon the institution that its success has been assured since the spring of 1890, and the president feels that the work he undertook to do ten years ago has been more than accomplished. The result already reached is far beyond his expectations, and much exceeds any hopes or dreams he dared to indulge in when he took charge of the institution.

At the present time the president can see almost unlimited opportunities for the future development and advancement of the Michigan College of Mines, which he would like to see improved.

The faculty of the Michigan College of Mines is composed of men who have had practical experience in manual as well as intellectual labor.

Every member of the faculty is a specialist in his line, an original investigator, and has published more or less in his department, while all except one have been connected with leading colleges and universities as teachers prior to taking up their labors here.

It would be impossible to keep such men upon the faculty here but by carrying out the principles stated above. It is the treatment given them that alone is retaining them, and not their salaries; for, when the expenses of living are considered, the professors of the Michigan College of Mines are paid much smaller salaries than are those of any of the other three State schools for higher education. This is one more proof that it is economy on the part of boards and presidents to treat the members of their faculties decently and properly. The president of the Michigan College of Mines has no hesitancy in saying that he regards an ordinary professorship in the State or any other similar university as a much better paid position than his, when the attendant expenses and cares are taken into consideration.

Yet with all these salary disadvantages there is not a single inefficient man in the faculty, but each one is hard working and energetic. I do not believe that, in proportion to their numbers, a better faculty can be found in the United States.

The conditions in this institution are so unlike those of any other in this country that each instructor is obliged to rearrange his work, and prepare in part, or as a whole, his own text-books, which have to be largely original. There are two special reasons why this must be done: First, because all other schools are giving instruction in other courses, hence their publications usually look towards training men in other lines, as well as in mining engineering; and, second, because most of the books are too theoretical, wanting the practical applications of the principles that they inculcate, while in the majority of cases they are padded with material that is of no use to an engineer. It would be a great gain to the Michigan College of Mines if the members of its faculty could be relieved of a large amount of the drudgery that they now have to do, but which could just as well be done by assistants at a moderate cost, thus enabling each instructor to have some opportunity to keep up with the progress in his departments, and to publish needed text-books and original investigations. Each professor has to do almost double the work done by the average professor in any college or university in the land, that I know of. They are interested and willing, but it is a most short-sighted policy on the part of the college to allow such a burden to be borne by them. The work done here now is done at the pace that kills; no men can stand it for many years and keep up at all with the times. The inevitable results are mental stagnation, breaking down and death. The faculty are alike overworked, and a relief must be found, and that quickly.

In comparing the number of students at the Michigan State University and the Michigan State Agricultural College, it ought always to be understood that the instruction given at the University and College is in a great measure general and appeals to the masses, while at the State Mining College it is more special and attracts only those engaged in some work connected with the mineral interests of the country. The comparison, then, more naturally falls with some single course, like those of civil and mechanical engineering, although the total number of students in the United States pursuing such subjects is far larger than those following mining studies. In 1842–43 the University started with 25 pupils and had but 64 in 1851–52, having in three years run down from 84 (the

largest number) to 64. It then commenced to rise, but its progress has by no means been uniform. Its upward jumps have been most marked in 1852–53 (222 students), 1855–56 (416), 1864–65 (856), 1866–67 (1235), 1890–91 (2153), 1892-93 (2788), 1895–96 (3014); but it has had more or less prolonged periods of depression, as from 1856 (453) to 1860 (429), 1867 (1255) to 1877 (1110). The Agricultural College, thirteen years after its organization, had only 79 students (1869–70).

The oldest scientific school in the United States—the Lawrence Scientific School of Harvard University—opened in 1847, never had in all its courses over 82 pupils at one time until 1891. It had 79 in 1865–66 and ran down to 14 in 1886–87, yet it has so gained since that this year there are 368 students.

In the table given on page 708 is placed the number of students in mining engineering for the last eleven years, so far as known, in most of the schools teaching mining engineering in the United States.

The Massachusetts Institute of Technology in reality has more students than are credited to it in this table, since the published lists give only those in the last three years of the course. The probability is that this school now has in all its classes as many students as Columbia. This year, it will be seen, that such famous institutions as Harvard, Columbia and Lehigh Universities have, respectively, 13, 44 and 37 students in mining engineering, while the Michigan College of Mines has 140.

The number of students taking instruction in mining engineering in the above two schools of mines it is impossible to ascertain. The numbers here given include for Colorado all students that have been enrolled as candidates for civil and electrical engineering and in chemistry. In the earlier years it also embraces others. In 1894–95 there were enrolled in the senior and junior classes 30 candidates for the mining engineering degree, and in the same classes for 1896–97, 47 candidates for the same degree. In 1894, 4 men were graduated in the mining engineering course, and in 1896, 9 men in the same course.

The Colorado State School of Mines was established in 1874, has more buildings, a larger income and a larger faculty than

Number of Students Enrolled in Mining Courses in America.

Name of School.*	1886-87.	1887-88.	1888-89.	1889-90.	1890-91.	1891-92.	1892-93.	1893-94.	1894-95.	1895-96.	1896-97.	Total Graduates.
University of California, 1877	17	24	24	25	30	32	24	52	39	60	99	63
Case School of Applied Science, 1891	.....	.....	.....	.....	.....	9	12	12	.....	7	9	6
Columbia University School of Mines, 1864	74	51	46	45	48	60	52	51	60	47	44	433
Harvard University, 1867	.....	.....	.....	.....	.....	.....	.....	.....	6	7	13	8
Lafayette College, 1866	.....	.....	.....	.....	16	.....	13	18	.....	5	.....	52
Lehigh University, 1866	58	56	61	68	52	68	64	51	58	43	37	92
Leland Stanford, Jr., University, 1890	.....	.....	.....	.....	.....	.....	.....	7	10	.....	.....	.....
Massachusetts Institute of Technology, 1865	16	12	14	18	17	17	29	22	33	32	24	143
University of Michigan, 1865	7	12	7	5	6	8	5	5	3	1	0	50
University of Minnesota, 1889	.....	.....	.....	.....	3	3	17	17	.....	22	34	5
College of Montana, 1888	.....	12	9	8	17	16	.....	21	12	.....	.....	9
University of Nevada, 1892	.....	.....	.....	.....	.....	.....	.....	19	23	4	.....	10
New Mexico School of Mines, 1893	.....	.....	.....	.....	.....	.....	.....	11	.....	23	4	.....
Ohio State University, 1879	9	11	18	21	17	21	36	20	.....	37	33	35
Pennsylvania State College, 1893	.....	.....	.....	.....	.....	.....	.....	9	22	23	23	7
South Dakota School of Mines, 1887	.....	.....	.....	.....	2	19	19	.....	6	15	.....	1
University of Tennessee, 1889	.....	.....	.....	3	3	3	7	5	4	6	.....	.....
Vanderbilt University, 1888	.....	.....	1	1	1	1	1	1	2	0	.....	1
Washington University, 1871	.....	.....	.....	13	15	11	11	4	.....	.....	.....	35
University of Wyoming, 1894	.....	.....	.....	.....	.....	.....	.....	.....	2	2	7	.....
Michigan College of Mines, 1886:												
Total enrollment	23	29	40	35	65	78	101	82	94	94	140	.....
New students	23	16	15	16	46	40	45	17	49	44	77	.....
Number of graduates	.....	7	6	5	4	0	8	17	22	18	7	94

* Date given after the name indicates the date of the establishment of the Mining Engineering Courses when known.

Number of Students Enrolled in all Courses in the Missouri and Colorado State Schools of Mines.

	Missouri State School of Mines.	Colorado State School of Mines.		Missouri State School of Mines.	Colorado State School of Mines.
1873–74	75	.....	1885–86	72	30
1874–75	107	.....	1886–87	46	.....
1875–76	101	.....	1887–88	52	45
1876–77	70	.....	1888–89	66	.....
1877–78	64	.....	1889–90	69	68
1878–79	43	.....	1890–91	80	.....
1879–80	71	.....	1891–92	83	91
1880–81	71	.....	1892–93	116	106
1881–82	96	.....	1893–94	121	121
1882–83	82	94	1894–95	96	130
1883–84	110	.....	1895–96	72	135
1884–85	71	23	1896–97	104	161

Graduates in all courses: Colorado State School of Mines..... 67
" " " Missouri State School of Mines..... 31

the Michigan College of Mines. It is supported by a one-fifth mill tax.

The people of Colorado take an interest in it, and its catalogue states that it is a "School of Applied Science," and, as such, no direct comparison can be made between it and the Michigan College of Mines, which only deals with questions relating to the winning of mineral wealth. The Colorado school, although the older, has in part modeled its instruction and its catalogue on the Michigan plan.

The Colorado School of Mines is situated not far from Denver, almost on the main line of travel, and in a district which is much better adapted to furnish students than the Michigan locality, particularly as it is a great resort for persons afflicted with lung troubles.

Its requirements for admission are much lower than are those of the Michigan College of Mines. The school is a good one and is doing good work. It certainly has a much better location for giving instruction in metallurgy than has the Michigan College, which latter, on the other hand, is much more favorably situated for giving instruction in mining and geology.

The Missouri School of Mines was organized in 1871, and

the number of students enumerated includes academic or preparatory students, candidates for degrees in civil engineering, chemistry and other subjects. It is not only a general "School of Applied Science," but an academy also. Its location is better for obtaining students than is that of the Michigan College of Mines, but is far less adapted to aid in teaching mining engineering than it is to assist in the instruction of agricultural students. There is a strong probability of its removal to another district.

Of the 390 students that have been enrolled in the Michigan College of Mines, 248 were residents of Michigan. The others came from Alabama, British Columbia, Nova Scotia, Ontario, Colorado, Cuba, District of Columbia, England, Germany, Idaho, Illinois, Indiana, Japan, Kansas, Massachusetts, Mexico, Minnesota, Mississippi, Missouri, Montana, New Hampshire, New Jersey, New York, Ohio, Oregon, Pennsylvania, Peru, Scotland, South Africa, South Dakota, Texas, Utah, Virginia, West Virginia, Wisconsin and Wyoming.

Of the 117 occupations followed by the fathers of these pupils, more were engaged in farming than in any other business. In fact the training given in engineering work at the Mining College is so broad that it offers one of the best fields now open to farmers' sons, and it is a wonder that more do not avail themselves of it.

The average age of the students is from 21 to 22 years, although at the present time there are over 25 men in the school of from 25 to 55 years of age. The college prides itself upon its usefulness to these practical and experienced men.

Number of Students from Michigan and Number from Other Districts.

	Michigan.			
	Upper Peninsula.	Lower Peninsula.	Entire State.	Outside of State.
1886–87, .	20	3	23	0
1887–88, .	20	6	26	3
1888–89, .	21	8	29	11
1889–90, .	18	7	25	10
1890–91, .	33	15	48	17
1891–92, .	34	23	57	21
1892–93, .	44	20	64	37
1893–94, .	38	20	58	24
1894–95, .	46	16	62	32
1895–96, .	46	12	58	36
1896–97, .	57	17	74	66

Amongst the difficulties with which the Michigan College of Mines has had to contend are: Its distance from large cities; its extreme northern locality, and the fact that it is not situated on one of the great lines of travel, and hence is much less known than it otherwise would be; and the general ignorance and misinformation in the southern portion of the State, and generally throughout the country, regarding the northern peninsula of Michigan, its people, its climate and its resources. The college, far more than any other factor, during the past few years, has been the means of dissipating much of the ignorance concerning this part of the country, and making it more widely and favorably known amongst a large class of people who are not naturally and directly reached by its vast mining operations.

Amongst the special advantages enjoyed by the college are:

1. The opportunities afforded by the kindness of the officers of the various mining companies in the copper, iron and gold districts, which enable the students to become familiar with the hoisting plants, timbering, and methods of working the deepest mines (4900 feet vertical) in the world, where the largest and most powerful machinery and stationary engines are employed.

2. Its unrivalled location for the study of crystalline rocks of both eruptive and sedimentary origin, which, in this Northern Peninsula of Michigan, show the most complicated and varied phenomena, including the basement-conglomerates of several geological formations.

3. That, so far as it goes, the equipment in each department in the school is modern, and has been selected expressly for purposes of instruction and not for show.

4. Its peculiar elective system.

5. The practical nature of the instruction, and the mining atmosphere surrounding the institution.

6. Its confinement to mining alone, and the great length of its college year.

Some Statistics of Engineering Education.

BY DR. M. E. WADSWORTH,* HOUGHTON, MICH.

(Lake Superior Meeting, July, 1897.)

THE chief value of a paper like this consists in its statistical tables, putting on record material useful to future inquirers.

The data here given have been compiled from time to time as far back as 1874, and subsequently brought up to date. Much of the data could only be procured by personal inspection of the catalogues, as for example those relating to the Lawrence Scientific School, which were taken at that time from the catalogue of Harvard University, on file in the University Library, by the present writer, then an instructor in that institution.

These tables of statistics were first prepared for publication by the present writer in 1888–1890 and published in his "Report to the Board of Control of the Michigan College of Mines" (Michigan Mining School) for 1886 to 1890. Later they were revised, enlarged and brought down to date in his reports to the same Board for 1890–92; and since that time they have been, at his request, revised, greatly enlarged and kept up to date by Mrs. Frances H. Scott, the efficient Librarian and Secretary of the Michigan College of Mines. Great credit is due to her for her devotion to this work and her untiring efforts to have the statistics complete as far as possible.

The accompanying running commentary is given simply as a string to hold these statistics together, and in order to call attention to a few points.

In compiling any tables like those accompanying this paper, one is confronted by serious difficulties, some of which render these results only approximate.

In order to insure greater correctness and fullness in the statistics to be published the writer's reports for 1890–92 were sent to the president of every college, university, and technical school in the country, so far as known. Accompanying this was a personal letter which asked of each one his aid in correcting any errors, and in giving fuller statistics. But relatively few answers were obtained, and subsequently from one to many letters have been sent to each institution not formerly reply-

* President of the Michigan College of Mines.

ing, but known to have engineering courses, in order to obtain the needed information, which is still imperfect.

Since there seems to be some misunderstanding regarding such statistics, it may be said, here, that no students who have not announced their choice are credited in these tables to any course in engineering; and further, that schools which give degrees in one department of engineering only, like the Stevens Institute of Technology, the Rensselaer Polytechnic Institute, and the Michigan College of Mines, have all their special students credited to that department. Special students, when there are several courses, can only be assigned to any particular engineering course when they have been designated as belonging to that course by the authorities of the institutions where they are studying. Accurate statistics of the kind attempted here are very hard to procure owing to the lack of records in many colleges, the disinclination of the authorities of many of them to have the facts known, the separation of the students into their special courses not taking place until the end of the first or second year, etc. The Sheffield Scientific School of Yale University has no classification of its students, and therefore no information can be given about it from the standpoint of this paper.

The earlier catalogues of Lafayette College were in this condition, but subsequent to a strong criticism made by the present writer upon this indiscriminate mingling of classical, scientific and engineering students (see his report for 1891–92, p. 12), an excellent catalogue has been published.

There seems to be no real reason why the differentiation of students should not be allowed upon their entrance, if proper provision were made for subsequent changes. This would enable a better statistical record to be kept, and it ought to be of great assistance to the instructors in making proper preparations for their subsequent work. Catalogues like those now published by Lehigh, Harvard, Lafayette, Columbia and some others are models in exhibiting the differentiation of the students. On the other hand, it would seem as if the majority of college catalogues were issued for no other purpose than to conceal the truth and cozen the public. Why cannot clear, distinct and truthful statements be made, showing the number of students in each course, the exact equipment, and other means of instruction, stating which of the members of the

corps of teachers are figure-heads, which are those with whom the student will come into personal contact, etc.

How long will it be possible for catalogues to dilate (as one of recent date has done) upon the extraordinary means of instruction that an engineering student would enjoy with such a remarkable equipment and so distinguished a faculty, when the equipment consisted of some chemical apparatus, and the entire faculty numbered only the president, who wrote the catalogue, and his assistant.

The labor of compiling and obtaining engineering statistics is immense—a labor the magnitude of which no one who has not undertaken it can realize. This paper alone has cost much more than a year of time and thousands of letters; and yet, in spite of every care taken, there will unquestionably be found grave mistakes in it, and many of the institutions will consider that injustice has been done them. This, however, if done, is not intentional, and when made known, it will be corrected as far as possible in future publications.

Table I. and part of Table II. give the number of students in the courses in engineering in the various schools in the United States as far as possible for each year since the course was established. These tables show that engineering education in the United States has been, on the whole, a thing of comparatively recent date—the pioneer schools being the Rensselaer Polytechnic Institute, established in 1824, the Lawrence Scientific School, dating from 1846, and the Sheffield School from 1847. But little further prominent work was done until 1863, when the Columbia School of Mines was established and followed rapidly by numerous other engineering schools.

Civil engineering has led in this country, and has had various periods of advance and depression. The advance was most marked in 1887–88 and in 1895–96, while 1896–97 has been a period of depression.

Mechanical engineering has, on the whole, progressed until the past year, which in most of the colleges has been a period of diminution in the number of students. The same thing has also occurred in the case of electrical engineering, which further suffers a natural reaction from having been greatly overdone.

In mining engineering 1874–75, 1883–85 and 1894–95 were periods of advance; the last having been followed, to the present date, by depression in most colleges, with notable excep-

tions in the cases of the University of California and the Michigan College of Mines, which have greatly increased their attendance during this college year. These are the only two engineering schools in this country that now have over 50 students in mining engineering. These remarks do not apply, however, to the well-known schools of mines situated respectively at Rolla, Mo., and Golden, Colo., for the reason that no information can be obtained from them giving the number of mining engineering students in each. These schools are quite fully discussed by the present writer in another paper (*ante*, pp. 707–710), to which any one interested in them is referred. In the same article there is published also a table giving the number of students enrolled in mining engineering courses during the past eleven years in the United States.

In order to enable one to correlate the chief schools of engineering in the United States, Table III. has been prepared, to show at a glance those institutions that have had fifty or more pupils in any course in engineering since the school year 1885–86.

The various statistics collected relating to engineering education seem to show that, as a rule, most of the schools in the United States run to specialties, usually one, or two, of the courses exceeding the others. Cornell University, however, all but leads in three of her engineering courses, being in mechanical engineering only slightly overtopped by the Stevens Institute.

Of the schools that give engineering courses, the Columbia School of Mines is less known from its mining engineering work than by its courses in architecture and civil engineering; Cornell University stands high for civil, electrical and mechanical engineering; the University of Illinois for civil and electrical engineering; Lehigh University for civil, electrical and mechanical engineering; the Massachusetts Institute of Technology is especially noted for its courses in architecture, electrical and mechanical engineering; the University of Michigan for civil, electrical and mechanical engineering; the Ohio State University for electrical engineering; Purdue University for electrical and mechanical courses; the Rensselaer Polytechnic for civil engineering; the University of Wisconsin for electrical and mechanical engineering courses; the Stevens Institute for mechanical engineering; Michigan Agricultural College and the Worcester Polytechnic for the same, and the Rose Polytechnic for its electrical engineering. (See Table III.)

In mining engineering the leading schools in the world, so far as shown from the records here published, are Freiberg, Leoben, Clausthal, Berlin, Paris, St. Etienne, Schemnitz, Przibram, Michigan College of Mines, California, Columbia, Lehigh, Massachusetts Institute of Technology, and Colorado. The Colorado school has been rapidly improving during the past few years, and, if its students were differentiated during the first two years, might be found to have more pupils taking the mining engineering course than any other school in the United States.

In addition to the question of yearly attendance in engineering courses in the United States, there is here given some comparison of the relative number of students in those of the foreign schools from which statistics have been obtained. The School of Mines at Clausthal (See Table III.), established in 1775, had 21 pupils in 1850–51, and not until 35 years (1885–86) later did it reach 100; while in 1870–71 there were only 20 students, and 73 was the largest number in any one year from 1850 to 1883. In 1896–97 there were 191 students. This is one of the leading schools of the German empire—only one other standing ahead of it—Freiberg.

The Berlin *Bergakademie* was founded in 1700 and reorganized in 1860 under the special auspices of the Prussian Government, while the Prussian Geological Survey was permanently attached to the school and the director of the *Bergakademie* placed in charge of both organizations. The government has endeavored by every means in its power to force this school to the front and to discriminate against the Clausthal one, but it has failed, owing to the unsuitable location. The buildings of the Berlin school are palaces, beside which the buildings of the Michigan College of Mines are hovels. In 1874–75 the Berlin institution had 93 pupils; in 1878–79, 123; in 1881–82, 92, when it ran up until it had 152 students in 1884–85, the largest number reported. In 1893–94 there were 119 students. (See Table III.)

The Freiberg *Bergakademie* was organized in 1765; but we have no statistics until 1845–46, when there were 48 students. The number was variable, falling to 46 and again rising to 92, until in 1857–58 it reached 129, and the next year 152. After that the number fell away, until in 1870–71 there were only 40 men in attendance. It again rose, reaching 114 in 1874–75, and 236 in 1896–97, which is the largest number reported. It

is to be remembered that this is the most famous mining school in the world and also now the largest, so far as known, except possibly Leoben. (See Table III.)

The Leoben *Bergakademie* was organized in 1840, had 128 students in 1886–87, 223 in 1893–94, and 213 in 1895–96. It has generally had more students than Freiberg.

The National School of Mines at Paris was established in 1778, but during the French Revolution it was discontinued and not re-established until 1816. The first statistics we have are for 1883–84, when there were 79; in 1890–91, 117, and in 1895–96, 139 pupils. For the decade from 1884 to 1894 the number of students has been remarkably uniform; lowest 98, highest number 139. The largest number in any entering class as given is 46, and the lowest number 17. In 1894–95 there were 43 entering pupils. (See Table III.)

Of other mining schools, that at St. Etienne had 106 pupils in 1886–87, but the next year only 78. In 1895–96 it had 80 students, and in 1896–97, 103; while the largest entering class given was 35, in 1879–80.

The University of Japan had, in 1893–94, 27 pupils in mining engineering, which is the largest number given; it had only 7 in 1886–87. (See Table IV.)

In looking over the various *Bergschulen* or elementary mining schools of the German and Austrian empires, that located at Bochum is one of the largest, so far as the records obtainable show. Between 1874 and 1893 the largest number of pupils was 184 in the years 1885–86 and 1891–92, while the lowest number was 93 in 1882–83. (See Table V.)

The other *Bergschulen* run in numbers from 7 to 55, averaging from 20 to 40 pupils each.

Table VI. gives the number of students graduated in the different mining engineering courses in the United States. Of these colleges Columbia leads in the number of her graduates, 433, followed by the Massachusetts Institute of Technology with 133, Lehigh University, 92; the Michigan College of Mines, 94; Colorado School of Mines, 67, and the University of California, 63.

If the number of students which graduated in 1896–97 were known, the relative standing might be changed in some cases.

TABLE I.—*Number of Students in the Courses in Engineering in the United States.*

Name of the School.	1863-64.	1864-65.	1865-66.	1866-67.	1867-68.	1868-69.	1869-70.	1870-71.	1871-72.	1872-73.	1873-74.	1874-75.	1875-76.	1876-77.	1877-78.	1878-79.	1879-80.	1880-81.	1881-82.	1882-83.	1883-84.	1884-85.	1885-86.	1886-87.	1887-88.	1888-89.	1889-90.	1890-91.	1891-92.	1892-93.	1893-94.	1894-95.	1895-96.	1896-97.
Alabama Polytechnic Institute:																																		
Civil	...	...	...	...	...	...	...	...	1	0	1	3	7	12	13	22	16	12	9	6	8	12	4	16	24	24	28	34	29	13	7	10	7	12
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13	34	38	39	44	55
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8	26	30	35	40	46
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	1	0	0	0	0	0	0	0	2	0	2	0	0	0	1	0
University of Alabama:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	19	14	.....	6	2
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	2	.....	4	4
Scientific	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	44	58	.....	51	36
University of Arizona:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	.....	.....	.....	.....
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	13	.....	.....	.....	.....
Armour Institute:																																		
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	20	21	21	.....
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	33	49	56	86	.....
Chemical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	6	6	.....	6	.....
University of California:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	30	38	38	34	49	42	53	52	57	58	83	56	50
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	17	17	22	24	28	35	30	58	84	108	128	133
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	19	17	17	24	24	25	30	32	24	32	39	60	110
***Case School of Applied Science:**																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	25	.....	30	.....	26	18
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	.....	41	.....	31	27
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	20	.....	38	.....	32	42
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9	12	12	.....	7	9
Colorado School of Mines: Civil, Electrical, Mining, Chemistry	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	94	.....	23	30	.....	45	.....	68	.....	91	106	121	130	135	161

* "Numbers for 1892-93 are about 14 per cent. less than for 1893-94; proportion in departments about the same."—Letter from President.

TABLE I.—*Continued.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
Colorado University:																																	
Civil	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	4	4	5
Electrical	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	6	17	22
***Columbia College School of Applied Science:**																																	
Civil	...	...	...	...	...							53	.....	.....	.....	61	25	21	23	29	44	59	66	86	84	82	78	87	100	92	78	84	67	72
Electrical	...	...	...	...	...							...	.....	.....	.....	...	...	...	...	...	...	...	...	...	...	...	19	21	26	41	87	113	117	134
Other Courses	...	...	...	...	...		68	76	.....	94		34	.....	.....	.....	141	164	142	132	126	85	93	81	100	95	104	124	144	164	115	108	102	139	143
Mining	...	43	79	...	...							58	.....	.....	.....	41	63	86	112	109	157	116	88	74	51	46	45	48	60	52	51	60	47	44
†Cornell University:																																	
Civil	...	...	...	...									82	70	58	43	37	38	.....	.....	.....	94	95	112	111	126	134	137	139	126	116	122	119	150
Mechanical	...	...	...	...		66	...	.....	121	...	...	...	56	54	44	33	30	33	.....	.....	.....	53	63	109	136	158	174	211	250	289	332	284	200	243
Electrical	...	...	...	...									.....	.....	.....	.....	.....	.....	.....	.....	.....	29	38	59	53	125	172	214	250	257	239	211	293	230
Delaware College:																																	
Civil	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	14	10	8
Electrical	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10	13	13
Mechanical	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5	6	4
‡Haverford College:																																	
Mechanical	...	...	...	...	...	...	...	.....	.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	14	12	11	.....	13
§University of Illinois:																																	
Civil	...	...	...	...	...	...	1	Not able to find	46	48	47	56	30	29	31	31	32	39	.....	.....	.....	.....	.....	.....	.....	.....	62	95	87	92	86	82	73	76
Electrical	...	...	...	...	...	...	...		.....	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1	29	94	123	108	123	90
Mechanical	...	...	...	...	...	...	45		33	40	25	56	28	29	31	34	37	34	.....	.....	.....	.....	.....	.....	.....	.....	74	78	88	79	67	86	76	58
Mining	...	...	...	...	...	...	...		1	2	2	...	2	2	3	2	3	5	.....	.....	.....	.....	.....	.....	.....	.....	6	5	6	7	2	.....	.....	.....

* From 1874–75 the figures indicating the numbers in "Other Courses" include the courses in Architecture and all others given in the school not enumerated above.

† "Summary of students not printed in catalogues for 1882–83–84."—Letter from University.

‡ "Do not make much of engineering work. For about eight years have had an average of 12 or 14 students."—Letter from College.

§ Have not been able to obtain data for years from 1881–82 to 1888–89, inclusive.

TABLE I.—*Continued.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88	1888–89.	1889–90.	1890–91.	1891 92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
University of Kansas:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	13	12	17	18	22	21	15	18	17	32	37	36	100	27	22
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	6	8	14	28	44	54	62		64	62
Lafayette College:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	33	26
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	34	30
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	16	.....	17	18	.....	5	3
Lehigh University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	57	71	71	100	130	148	124	123	116	149	151	144	128	105	84
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13	8	12	23	41	75	91	118	145	141	136	108	85
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	32	47	72	67	79	91	81	66	63	92	105	115	99	108	96
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	47	63	75	68	58	56	61	68	52	68	64	51	58	43	37
Leland Stanford, Jr., University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	63	60	40	38
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	78	66	55
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	98	39	82	19
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	7	10	.....	.....
***Massachusetts Institute of Technology:**																																	
Civil	...	...	...	...	...	...	...	40	54	34	66	53	37	25	18	15	10	12	21	29	44	45	50	71	71	79	81	81	75	78	121	126	99
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	18	30	41	52	61	74	91	105	108	105	112	140	163	146	106	
Mechanical	...	...	...	...	...	...	...	5	9	17	34	31	37	21	17	14	20	19	33	54	74	75	89	100	99	95	104	102	103	95	178	179	117
Mining	...	...	...	...	...	...	...	12	6	9	30	25	26	15	12	15	14	12	27	28	26	19	16	12	14	18	17	17	29	22	33	32	24
Michigan College of Mines:																																	
Mining	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	23	29	40	35	65	78	101	82	94	94	140
Michigan State Agricultural College:																																	
Mechanical	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	34	57	60	93	113	127	114	119	122	121	128	120

* Statistics cannot be obtained for years prior to 1872. Freshmen students not classified.

TABLE I.—*Continued.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
University of Minnesota:																																		
Civil	...	...	...	...	...	...	...	2	3	3	4	7	5	3	1	2	1	.....	5	4	5	2	2	6	14	17	19	30	25	33	34	30	33	28
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	6	29	45	59	50	50	78	62
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	2	1	1	.....	20	21	65	76	86	43	117	77	69	43	101	57	15	22	24	27
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	3	17	17	19	22	34
*Missouri School of Mines:																																		
Civil, Chemistry and Metallurgy, Mining, Preparatory, etc.	...	...	...	...	...	...	...	...	...	...	75	107	101	70	64	43	71	71	96	82	110	71	72	46	52	66	69	80	83	116	121	96	72	104
Mississippi Agricultural and Mechanical College:																																		
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	61	.....	73	57	22
†College of Montana:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Chemistry	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	.....	.....
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	9	8	17	16	.....	21	12	.....	.....
‡University of Nebraska:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	86	46	20
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	86	52	80
University of Nevada:																																		
Mechanic'l and Agricultur'l	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5	6	.....	.....
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	19	23	4	.....

* Records of school do not give the number of students in each course.

† No catalogue issued for 1892–93; hence no record of E. M. students for that year.

‡ "No record kept of students in various engineering groups."—Letter from University.

TABLE I.—*Continued.*

Name of the School.	1863-64.	1864-65.	1865-66.	1866-67.	1867-68.	1868-69.	1869-70.	1870-71.	1871-72.	1872-73.	1873-74.	1874-75.	1875-76.	1876-77.	1877-78.	1878-79.	1879-80.	1880-81.	1881-82.	1882-83.	1883-84.	1884-85.	1885-86.	1886-87.	1887-88.	1888-89.	1889-90.	1890-91.	1891-92.	1892-93.	1893-94.	1894-95.	1895-96.	1896-97.
*College of New Jersey:																																		
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	13	10	14	22	27	33	40	37	33	37	44	55	55	67	93	152	160	156	109	85
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10	37	48	28	22	12	7	10
New Mexico School of Mines:																																		
Mining	...	...	...	...	..	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	11	}	23	{4
Chemistry	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	}		9
†Ohio State University:																																		
Civil	...	...	...	...	..	...	...	...	...	...	.....	.....	.....	.....	41	34	40	19	25	55	60	27	29	28	24	20	27	38	36	59	72	70	78	64
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	33	65	125	116	138	120
Mechanical	...	...	...	...	...	...	...	...	..	...	.....	.....	.....	.....	.....	.....	24	21	22	31	31	32	37	49	18	17	11	22	11	18	30	37	41	35
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	5	3	9	10	8	14	17	8	16	9	11	18	21	17	21	36	20	.....	37	33
Pennsylvania State College:																																		
Civil	...	...	...	...	...	...	...	..	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	6	3	4	1	5	15	17	28	37	36	37	18	30	31	33
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10	11	17	22	40	57	68	64	65
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10	18	22	21	19	31	44	44	50	45	38
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9	22	23	23
University of Pennsylvania:																																		
Civil	9	6	5	6	17	23	29	35	43	98	126	116	126	105	120	132	133	143	171	201	239	225	51	.....	.....	32	.....	30	32	36	34	56	45	34
Scientific and Chemistry																							105	198	170	140	203	134	153	164	71	100	98	81
Mechanical																							34	.....	.....	20	.....	40	68	82	140	127	78	104
Mining																							18	.....	.....	8	.....	6	6	.....	.....	1	.....	.....
‡Electrical																							.....	.....	.....	.....	.....	.....	.....	.....	.....	8	36	.....

* Numbers given in Electrical Engineering for 1892–93 and 1893–94 are for two upper years only; candidates for E. E. not asked to indicate it until they have finished three years (and from 1893–94, four years) in the B. S. or C. E. course.

† Previous to 1884 data approximated from annual reports.

‡ Number given as taking the Mechanical course in 1896-97 includes students taking the Electrical course.

TABLE I.—*Continued.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
*Purdue University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	34	52	70	.....	68	57	68	80
Electrical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	85	75	122	.....	163	173	153	120
Mechanical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	49	67	90	.....	118	108	133	142
Rose Polytechnic Inst.:																																	
Civil	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	3	4	16	22	28	20	24	20	26	23	20	25	20
Electrical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	81	107	131	133	139	121	91	76	80	66
Mechanical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	25	40	64	63	62	2	2	1	1	1	2	2	18	8	16
Chemical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	2	0	1	4	4	7	5	3	15	17	11	10	7
S. Dakota School of Mines:																																	
Civil, Mining, General Scientific	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	19	19	.....	6	55	51
Stevens Inst. of Techn'l'gy:																																	
Mechanical	...	...	...	...	...	...	...	...	49	61	89	95	79	89	77	74	100	132	152	178	168	176	176	185	196	196	213	210	264	264	256	263	254
†University of Tennessee:																																	
Civil	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	33	11	20	12	11	8	8	42
Electrical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5	4	35
Mechanical	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13	6	9	15	8	11	11	15
Mining	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	3	3	7	5	4	6	.....
University of Texas:																																	
Civil	...	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	16	18	18	21	.....	21	16	19	25

* Classification for previous years cannot be obtained from the University records.

† Civil and Mechanical courses since 1870–71, but records from that time to 1888–89 are incomplete. Mining course dropped in 1896. Electrical course now known as Physics and Electrical Engineering.

TABLE I.—*Continued.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
Tufts College:																																	
Civil	...	...	...	...	1	7	5	3	...	8	.....	7	12	13	16	12	6	4	6	} 8	10	24	24	24	20	22	38	.....	57	64	89	} 91	80	76
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
University of Utah:																																	
*Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	9	13	28	3	.....
General Science	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
†Vanderbilt University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	20	23	29	27	37	37	19	34	51	42	40	45	36	33	32	8	8
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	6	6
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1	2	4	5	8	9	14	14	8	7	4
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1	1	1	1	1	1	2	.....	.....
University of Vermont:																																	
Civil, Electrical, Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	28	.....	41	53	42	45	58
‡Washington University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8	9	9	11	4	9	10	8
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	3	7	6	3
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9	11	8	5	7	7	7	7
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13	15	11	11	4	.....	.....	.....
Western University:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	36	33	.....	27	32	22	.....	28
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21	34	42	.....	36
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	10	30	.....	40	30	26	.....	26
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5

* Discontinued for lack of funds.

† "Since 1886, when courses for M. E. and E. M. were established, I am not quite sure of the accuracy of the relative divisions; it is not very wide of the truth."—Letter from Secretary of Engineering Faculty.

‡ Students not classified in catalogue until junior year.

TABLE I.—*Concluded.*

Name of the School.	1863–64.	1864–65.	1865–66.	1866–67.	1867–68.	1868–69.	1869–70.	1870–71.	1871–72.	1872–73.	1873–74.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.	1896–97.
University of West Virginia:																																	
Civil and Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	...}	8	11	11	47	33	87	75
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	...}			4	22	25	145	78
*University of Wisconsin:																																	
Civil	...	...	...	...	...	...	...	...	...	...	.....	11	12	9	7	8	11	11	13	14	14	12	6	17	18	29	27	38	41	45	54	49	45	66
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	6	6	7	6	9	24	.....	9	15	52	56	87	101	97	92
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	1	2	3	8	15	17	14	11	24	30	31	43	47	42	33	51	66	51	57
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	3	1	2	6	5	3	2	2	3	2	2	2	3	3	1	.....	.....	.....	.....	1	.....
†Worcester Polytechnic Inst.:																																	
Civil	...	...	...	...	...	...	...	35	41	37	51	36	29	25	18	13	14	15	20	22	23	21	20	18	27	29	33	26	28	26	23	16	16	26
Electrical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3	3	3	16	25	31	32	22
Mechanical	...	...	...	...	...	...	...	33	37	31	52	51	58	51	53	62	61	61	88	95	88	94	110	100	115	116	73	88	119	132	167	134	129	130
University of Wyoming:																																	
Mechanical	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	12	15	7	12
Mining	...	...	...	...	...	...	...	...	...	...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	2	7

* Students not classified until 1874–75. Course in Mining Engineering discontinued.

† First class enrolled in 1868.

NOTE.—Have been unable to obtain any data from Swarthmore College, Union University, Virginia Agricultural and Mechanical College, University of Virginia, Washington and Lee University, Yale University and McGill University, all of which have one or more courses in engineering.

TABLE II.

Year.	Total Number of Students Enrolled.	UNIVERSITY OF MICHIGAN.* Graduates.													LAWRENCE SCIENTIFIC SCHOOL. Number of Students in Courses in Engineering, etc.						MICHIGAN AGRICULTURAL COLLEGE.†		KENSSELAER POLYTECHNIC INSTITUTE.	LEOBEN BERGAKADEMIE.***	PRZIBRAM BERGAKADEMIE.†††	SCHEMNITZ BERGAKADEMIE.‡‡‡ Number of Students Enrolled.		FREIBERG (SAXONY) BERGAKADEMIE.‡	PARIS NATIONAL SCHOOL OF MINES.§		ST. ETIENNE SCHOOL OF MINES.‖		CLAUSTHAL BERGAK'D'MIE.¶	BERLIN BERGAKADEMIE.**	BALLARAT (AUSTRALIA) SCHOOL OF MINES.††
		Literature, Science and Arts.	Medicine and Surgery.	Law.	Pharmacy.	Homœopathy.	Dentistry.	Total Number of Graduates.	Civil.	Mining.	Mechanical.	Electrical.	Civil.	Mining.	Other Courses.	Mechanical.	Electrical.	Mining.	Forestry.	Number of Students Enrolled.	Number of Students enrolled.	Number of Graduates.	Number of Students enrolled.	Number of Students enrolled.	Number of Students enrolled.	Mining.	Forestry.	Number of Students enrolled.	Number of Graduates.	Number of Students enrolled.	Number of Graduates.	Number in Entering Class.	Number of Students enrolled.	Number of Graduates.	Number in Entering Class.	Number of Students enrolled.	Number of Students enrolled.
1842–43	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1843–44	23	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1844–45	53	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1845–46	52	12	.....	.....	.....	.....	.....	12	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	46	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1846–47	70	12	.....	.....	.....	.....	.....	12	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	53	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1847–48	88	12	.....	.....	.....	.....	.....	12	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	46	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1848–49	84	16	.....	.....	.....	.....	.....	16	.....	.....	.....	.....	.....	.....	8	.....	.....	.....	.....	61	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1849–50	72	24	.....	.....	.....	.....	.....	24	.....	.....	.....	.....	.....	.....	18	.....	.....	.....	.....	59	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1850–51	61	12	.....	.....	.....	.....	.....	12	.....	.....	.....	.....	.....	.....	31	.....	.....	.....	.....	60	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1851–52	222	16	.....	.....	.....	.....	.....	16	.....	.....	.....	.....	.....	.....	66	.....	.....	.....	.....	72	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1852–53	224	37	6	.....	.....	.....	.....	43	.....	.....	.....	.....	.....	.....	69	.....	.....	.....	.....	78	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1853–54	228	45	27	.....	.....	.....	.....	72	.....	.....	.....	.....	.....	.....	48	.....	.....	.....	.....	88	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1854–55	416	62	34	11	.....	.....	.....	107	.....	.....	.....	.....	.....	.....	54	.....	.....	.....	.....	70	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1855–56	483	40	41	21	.....	.....	.....	102	.....	.....	.....	.....	.....	.....	64	.....	.....	.....	.....	79	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1856–57	483	50	23	17	.....	.....	.....	90	.....	.....	.....	.....	.....	.....	56	.....	.....	.....	.....	92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1857–58	499	62	30	20	.....	.....	.....	112	.....	.....	.....	.....	.....	.....	67	.....	.....	.....	.....	129	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1858–59	450	49	27	20	.....	.....	.....	96	.....	.....	.....	.....	.....	.....	69	.....	.....	.....	.....	102	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1859–60	429	89	29	24	.....	.....	.....	142	2	.....	.....	.....	.....	.....	62	.....	.....	.....	.....	143	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1860–61	527	86	34	21	.....	.....	.....	141	6	.....	.....	.....	.....	.....	61	.....	.....	.....	.....	187	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1861–62	671	53	48	24	.....	.....	.....	125	6	.....	.....	.....	.....	.....	57	.....	.....	.....	.....	148	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1862–63	615	47	39	44	.....	.....	.....	130	7	.....	.....	.....	.....	.....	56	.....	.....	.....	.....	106	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1863–64	602	32	48	44	.....	.....	.....	124	.....	.....	.....	.....	.....	.....	75	.....	.....	.....	.....	104	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1864–65	806	26	51	71	.....	.....	.....	148	.....	.....	.....	.....	.....	.....	72	.....	.....	.....	.....	105	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1865–66	903	32	71	80	.....	.....	.....	183	.....	.....	.....	.....	.....	.....	79	.....	.....	.....	.....	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1866–67	1285	44	74	109	.....	.....	.....	227	9	.....	.....	.....	.....	.....	80	.....	.....	8	.....	92	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1867–68	1255	42	82	146	.....	.....	.....	270	9	.....	.....	.....	.....	8	49	.....	.....	.....	.....	96	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

1868-69	1223	55	81	152	......	......	......	298	11	......	......	11	......	......	41	......	......	82	10	147	......	......	......	......	77	......	......	......	......	......	......	......	23	......	......
1869-70	1114	44	98	129	23	......	......	294	......	......	......	12	......	......	48	......	......	79	10	185	......	......	......	......	74	......	......	......	......	......	......	......	27	......	......
1870-71	1112	77	86	120	28	......	......	311	14				......	......	85	......	......	129	12	145	......	......	......	......	40	......	......	......	......	......	......	......	20	......	......
1871-72	1110	66	82	117	21	......	......	286	14				......	......	27	......	......	141	12	151	......	......	......	......	69	......	......	......	......	......	......	......	29	......	10
1872-73	1207	85	89	142	5	......	......	321	83				......	......	40	......	......	131	5	199	......	......	......	......	76	......	......	......	......	......	......	......	36	......	16
1873-74	1163	78	91	123	9	......	......	301	29				......	......	42	......	......	143	15	209	......	......	......	......	86	......	......	......	......	......	......	......	54	......	41
1874-75	1105	76	71	127	20	......	......	294	65	......	......	4	......	......	29	......	......	121	21	181	......	......	......	......	114	......	......	......	......	......	......	......	50	98	81
1875-76	1191	101	79	186	18	......	......	334	35	......	......	10	......	......	34	......	......	156	15	186	......	......	......	......	139	......	......	......	......	......	......	......	73	93	68
1876-77	1127	76	93	159	81	......	9	368	38	......	......	18	13	......	16	......	......	164	17	166	......	......	......	......	165	......	......	......	......	......	......	......	52	111	57
1877-78	1110	76	84	122	28	13	10	333	39	......	......	13	6	......	16	......	......	154	15	160	......	......	......	......	156	9	......	......	......	......	......	......	45	118	64
1878-79	1230	73	98	148	22	22	14	377	26	......	......	10	8	......	9	......	......	239	30	126	......	......	......	......	160	11	......	18	......	......	......	......	45	123	74
1879-80	1372	60	104	193	25	25	15	422	8	......	......	3	5	......	11	......	......	232	19	104	......	......	......	......	145	18	......	13	26	......	19	85	53	108	66
1880-81	1427	66	92	175	24	19	34	410	6	......	......	......	4	......	33	......	......	264	6	108	......	......	......	......	122	12	......	19	20	......	......	......	56	111	90
1881-82	1534	91	101	145	33	23	37	430	......	......	......	......	6	......	24	......	......	221	33	172	......	......	......	......	119	12	......	10	17	......	......	......	68	92	101
1882-83	1534	78	91	170	40	16	32	427	43	......	4		6	......	19	......	......	216	28	218	......	......	......	......	136	29	......	......	......	......	......	......	72	110	110
1883-84	1440	85	117	155	37	17	26	437	57	......	14		7	......	19	......	......	185	30	204	......	......	......	......	147	16	79	......	......	......	......	......	83	142	196
1884-85	1377	89	85	134	38	20	25	391	......	......	......	......	8	......	20	......	......	171	30	234	......	......	......	......	163	10	98	12	27	......	10	30	82	152	187
1885-86	1295	80	80	136	26	6	28	356	57	15		8	1	......	21	......	......	173	30	185	......	......	......	......	104	30	109	16	84	80	17	29	119	130	211
1886-87	1401	92	83	116	23	17	30	361	57	28		7	2	......	12	......	......	296	27	164	128	13	60	364	153	29	101	24	32	106	5	30	100	145	255
1887-88	1572	98	81	154	29	13	27	402	60	32		12	4	......	16	......	......	323	22	151	......	......	......	......	161	30	108	18	35	78	26	30	113	140	312
1888-89	1667	124	65	146	23	13	38	409	68	40		7	1	......	33	......	1	312	32	167	......	......	47	320	......	......	110	23	40	96	21	26	129	107	329
1889-90	1882	105	82	147	41	21	34	430	76	53		5	9	......	50	......	6	340	45	174	115	11	88	259	......	......	113	18	31	97	24	25	150	105	381
1890-91	2153	130	89	212	33	22	38	524	78	65		6	4	......	78	......	6	369	32	189	142	9	42	208	170	27	117	33	35	76	22	26	153	104	397
1891-92	2692	155	103	266	81	18	29	602	96	73		8	10	......	100	......	8	360	34	185	162	15	51	179	178	23	114	26	38	80	14	30	155	114	398
1892-93	2778	174	116	294	29	18	39	670	93	80	90	5	40	......	110	......	31	345	26	206	183	23	64	131	193	22	109	23	29	82	21	......	145	108	453
1893-94	2659	213	47	327	22	20	54	683	92	99	93	5	57	......	159	8	56	355	44	188	223	24	64	105	169	28	113	23	46	86	21	3	148	119	433
1894-95	2864	244	64	302	23	9	60	702	96	79	130	3	44	6	168	27	63	405	28	165	214	22	64	96	171	33	134	31	43	86	26	30	154	......	......
1895-96	3014	272	69	313	19	1	47	721	87	86	125	1	41	7	186	41	65	393	39	135	213	58	81	84	200	23	139	29	......	80	22	30	177	......	......
1896-97	2975	280	52	348	23	6	60	769	50	52	87	......	32	13	209	70	44	425	33	137	......	......	......	......	236	......	139	29	......	103	......	......	191	......	......

* Course of instruction established in 1841. † Course of instruction established in 1857.

‡ School founded in 1765. § School established in 1778. Disestablished during the French Revolution. Re-established in 1816.

‖ School established in 1678. Re-established in 1834. ¶ School established in 1775. Statistics from letter from the Director of the Bergakademie.

** School established in 1700. Statistics from *Preussische Zeitschrift für das Berg-Huetten-und Salinenwesen.*

†† School founded in 1871. The figures given include the students in all the courses, which are, at present, Civil, Electrical and Mechanical Engineering Geology, Architecture and Metallurgy.

*** Established in 1840 at Vordernberg. Moved to Leoben in 1849. ††† Established in 1849. Reorganized in 1874.

‡‡‡ Established in 1760.

TABLE III.—*Number of Students in Engineering Courses in the United States in which there were Fifty or More during the Past Eleven Years.*

	1886-87.	1887-88.	1888-89.	1889-90.	1890-91.	1891-92.	1892-93.	1893-94.	1894-95.	1895-96.	1896-97.
Civil Engineering.											
University of California	38	34	49	42	53	52	57	58	83	56	50
Columbia College School of Appl'd Science	86	84	82	78	87	100	92	78	81	67	72
Cornell University	112	111	126	134	137	139	125	116	122	119	150
University of Illinois	.....	.....	.....	62	95	87	92	86	82	73	76
Lawrence Scientific School	.....	.....	.....	.....	.....	.....	40	57	44	41	32
Lehigh University	130	148	124	128	116	149	154	144	128	105	84
Leland Stanford, Junior, University	.....	.....	.....	.....	.....	.....	.....	63	60	40	38
Massachusetts Institute of Technology	45	50	71	71	79	81	75	78	121	126	99
University of Michigan	57	60	68	76	78	96	93	92	96	87	50
College of New Jersey	37	44	55	55	67	93	152	160	156	109	85
Ohio State University	28	24	20	27	38	36	59	72	70	78	64
Purdue University	.....	.....	.....	34	52	70	.....	68	57	68	80
Rensselaer Polytechnic Institute	164	151	167	174	189	185	206	188	165	135	137
University of West Virginia	.....	.....	.....	.....	.....	.....	11	42	33	87	75
University of Wisconsin	17	18	29	27	38	41	45	54	49	45	66
Electrical Engineering.											
Alabama Polytechnic Institute	.....	.....	.....	.....	.....	13	34	38	39	44	55
Armour Institute	.....	.....	.....	.....	.....	.....	33	49	56	86	.....
Columbia College School of Appl'd Science	.....	.....	.....	19	21	26	41	87	113	117	134
Cornell University	38	59	83	125	172	214	250	239	211	293	230
University of Illinois	.....	.....	.....	.....	1	29	94	123	108	123	90
University of Kansas	.....	6	8	14	28	44	54	62	.....	64	62
Lawrence Scientific School	.....	.....	1	6	6	8	31	56	63	65	44
Lehigh University	12	23	41	75	91	118	145	141	136	103	85
Leland Stanford, Junior, University	.....	.....	.....	.....	.....	.....	.....	.....	78	66	55
Massachusetts Institute of Technology	61	74	91	105	108	105	112	140	163	146	106
University of Michigan	.....	.....	.....	.....	.....	.....	90	93	130	135	87
University of Minnesota	.....	.....	2	6	29	45	59	50	56	78	62
University of Nebraska	.....	.....	.....	.....	.....	.....	.....	.....	86	82	80
Ohio State University	.....	.....	.....	.....	.....	33	65	125	116	138	120
Pennsylvania State College	.....	.....	10	11	17	22	40	57	68	64	65
Purdue University	.....	.....	.....	35	75	122	.....	183	173	153	120
Rose Polytechnic Institute	.....	81	107	131	133	139	121	91	76	80	66
University of Tennessee	.....	.....	.....	.....	.....	.....	.....	.....	5	4	85
University of Wisconsin	9	24	.....	9	15	52	56	87	101	97	82
Mechanical Engineering.											
University of California	17	22	24	28	35	30	58	84	108	128	133
Cornell University	63	109	136	158	174	211	250	232	284	200	243
University of Illinois	.....	.....	.....	74	78	8	79	67	86	76	56
Lawrence Scientific School	.....	.....	.....	.....	.....	.....	.....	8	27	41	70
Lehigh University	79	91	81	66	63	92	105	115	99	103	96
Leland Stanford, Junior, University	.....	.....	.....	.....	.....	.....	.....	98	39	32	19
Massachusetts Institute of Technology	89	100	99	90	104	102	103	95	178	179	117
Michigan State Agricultural College	57	60	93	113	127	114	119	122	121	128	120
University of Michigan	28	32	40	58	65	73	80	99	79	86	52
University of Minnesota	21	65	76	80	43	101	57	15	22	24	27
Mississippi Agricul. and Mech. College	.....	.....	.....	.....	.....	.....	61	.....	73	57	22
University of Pennsylvania	.....	.....	20	.....	40	68	82	140	127	78	104
Pennsylvania State College	10	18	22	21	19	31	44	44	50	45	38
Purdue University	.....	.....	.....	49	67	90	.....	118	108	133	142
Stevens Institute of Technology	176	176	185	196	213	210	264	264	256	268	254
University of West Virginia	.....	.....	.....	.....	.....	.....	4	22	25	145	78
University of Wisconsin	24	30	31	43	47	42	33	51	66	51	57
Worcester Polytechnic Institute	100	115	116	73	88	119	132	167	134	129	130
Mining Engineering.											
University of California	17	24	24	25	30	32	24	32	39	60	110
Columbia College School of Mines	74	51	46	45	48	60	52	51	60	47	44
Lehigh University	58	56	61	68	52	68	64	51	58	43	37
Michigan College of Mines	23	29	40	38	65	78	101	82	94	94	140

TABLE IV.—*Number of Students in the Courses in Engineering in Some Foreign Schools.*

Name of the School.	1880-81.	1881-82.	1882-83.	1883-84.	1884-85.	1885-86.	1886-87.	1887-88.	1888-89.	1889-90.	1890-91.	1891-92.	1892-93.	1893-94.	1894-95.	1895-96.	1896-97.
Kingston School of Mining:																	
Mining	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	*54	.....	40
Laval University:																	
Civil	.....	.....	.....	.....	.....	.....	.....	.....	.....	5	5	4	3	14	15	18	20
Toronto School of Practical Science:																	
Civil	15	16	19	36	41	51	47	56	56	57	67	71	57	34	25	13	26
Electrical	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	17	35	56	71	63	46	53
Mechanical	.....	.....	.....	.....	.....	4	8	4	4	10							
†Mining	.....	.....	.....	.....	.....	.....	.....	.....	1	.....	.....	5	3	7	14	10	18
University of Japan:																	
Civil	.....	.....	.....	.....	.....	.....	32	40	36	34	39	45	54	60	65	.....	.....
Electrical	.....	.....	.....	.....	.....	.....	7	8	7	10	11	14	15	23	38	.....	.....
Mechanical	.....	.....	.....	.....	.....	.....	5	8	9	10	10	15	20	32	41	.....	.....
Mining	.....	.....	.....	.....	.....	.....	7	8	8	15	15	18	21	27	37	.....	.....

* Of this number, 22 are students taking prospectors' evening lectures only.

† Includes Metallurgy and Assaying.

TABLE V.—*Number of Students in Foreign Bergschule (Mining Schools).*

Name of the School.	1874–75.	1875–76.	1876–77.	1877–78.	1878–79.	1879–80.	1880–81.	1881–82.	1882–83.	1883–84.	1884–85.	1885–86.	1886–87.	1887–88.	1888–89.	1889–90.	1890–91.	1891–92.	1892–93.	1893–94.	1894–95.	1895–96.
These schools have a Mining Engineering Course only.																						
*Tarnowitz Bergschule	25	45	41	35	26	32	34	31	28	27	44	40	38	47	46	46	44	41	55	57	.....	.....
*Waldenburg Bergschule	20	19	21	21	21	18	24	22	17	13	12	20	17	20	18	17	18	20	20	22	.....	.....
*Eisleben Bergschule	28	31	29	33	20	21	37	37	44	50	48	44	34	38	43	33	39	33	33	37	.....	.....
*Bochum Bergschule	106	116	121	121	108	108	98	94	98	111	134	184	174	159	162	167	176	184	172	162	.....	.....
*Essen Bergschule	40	40	29	30	30	30	25	25	25	28	30	28	28	27	27	31	36	36	35	34	.....	.....
*Saarbrucken Bergschule	11	11	12	12	14	11	11	12	12	13	12	13	12	11	14	14	14	11	7	9	.....	.....
*Siegen Bergschule	42	43	38	33	34	14	14	30	33	33	33	32	29	25	26	30	36	38	40	36	.....	.....
*Bardenberg Bergschule	20	24	22	7	12	10	13	13	13	12	12	19	18	25	25	24	25	30	29	11	.....	.....
*Clausthal Bergschule	28	25	25	28	31	31	25	19	19	20	21	24	26	26	26	26	26	26	25	25	.....	.....
*Dillenburg Bergschule	22	22	12	9	7	16	14	10	9	18	14	21	17	18	15	20	16	19	20	19	.....	.....
*Wetzlar Bergschule	30	17	14	9	10	8	12	24	18	15	14	10	8	8	12	11	11	13	14	14	.....	.....
†Leoben Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	30	29	30	28	.....	36	36	36	39	40
†Klagenfurt Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9	9	12	10	10	10	15	13	14	14
†Mährisch Ostrau Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	25	23	28	25	.....	.....	32	28	54	48
†Dux Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24	17	16	10	.....	.....	82	28	33	25
†Przibram Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	41	40	39	36	36	32	31	31	32	31
Schemnitz Bergschule	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	25	25	25	23	40	41

* See *Prussische Zeitschrift für das Berg-Hütten-und Salinenwesen.*

† See *Oesterreichische Zeitschrift für Berg-und Hüttenwesen.*

TABLE VI.—*Number of Graduates Each Year in Mining Engineering Courses.*

Name of the School.	1864.	1865.	1866.	1867.	1868.	1869.	1870.	1871.	1872.	1873.	1874.	1875.	1876.	1877.	1878.	1879.	1880.	1881.	1882.	1883.	1884.	1885.	1886.	1887.	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	1897.	Totals.
Alabama Polytechnic Inst.	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	1	0	1	0	0	...	2
University of Alabama	...	...	...	...	...	...	..	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	0	2	...	...	...	...	2
University of California	...	...	...	...	...	...	...	...	...	...	...	...	...	7	2	9	4	2	2	0	1	4	3	5	1	3	1	5	4	5	...	4	1	5	68
Case School of Appl'd Science	...	...	...	...	...	...	..	...	...	...	...	...	...	...	...	..	...	...	...	...	...	†	...	...	...	...	1	0	3	0	1	0	1	...	6
Colorado School of Mines	...	...	...	...	...	...	...	...	...	...	*	...	2								6						4	9	9	6	4	18	9	...	67
Columbia Col. Sch. of Mines	*	...	...	13	20	11	9	6	4	5	8	12	17	25	23	20	15	16	20	20	37	36	16	23	5	12	6	5	9	8	5	13	14	...	433
Harvard University	...	...	...	*	...	...	4	0	0	2	1	1	0	0	0	Course dropped.					...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	8
University of Illinois	...	...	...	...	*	...	...	0	0				5								0	0	0	0	0	0	0	0	0	...	...	...	...	...	5
Lafayette College	...	...	*	...	...	0	2	3	3	2	2	2	5	3	3	0	3	0	1	1	2	1	2	2	0	0	2	...	1	4	4	2	2	...	52
Lehigh University	...	...	*	...	...	0	0	1	0	0	0	1	2	0	0	0	5	1	3	6	4	1	7	8	3	7	4	0	7	1	3	14	14	...	92
Leland Stanford, Jr., Univ.	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	0	0	...	...	...	...	...	.....
Mass. Institute of Technology	...	*	...	...	6	0	2	5	5	3	1	6	7	8	2	3	3	6	5	4	3	8	7	8	3	5	3	4	4	5	4	3	10	..	123
Michigan College of Mines	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	...	7	6	5	4	0	8	17	22	18	7	24
University of Michigan	...	*	...	2	6	7	8	1	1	0	0	0	0	0	4	3	3	0	0	0	0	0	2	1	1	4	2	0	2	2	1	0	1	...	46
University of Minnesota	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	...	0	0	0	...	2	3	...	5
Missouri School of Mines	...	...	...	...	...	...	...	*	...	...	1	1	3	2	2	1	2	0	1	1	4	2	1	2	0	0	0	1	1	2	1	2	2	...	32
College of Montana	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	...	...	2	2	0	4	1	...	...	9
University of Nevada	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	1	2	3	...	4	...	10
Ohio State University	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	1	1	1	1	2	1	2	3	2	4	2	0	0	1	1	1	6	6	...	35
University of Pennsylvania	...	...	...	...	...	...	...	...	...	...								47								1	0	1	0	...	...	...	...	...	49
Pennsylvania State College	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	1	2	4	...	7
South Dakota School of Mines	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	*	0	0	1	...	...	...	...	...	...	...	1
Vanderbilt University	...	...	...	...	...	...	...	...	...	†	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	0	1	0	...	...	...	...	1
Washington University	...	...	...	...	...	...	...	*	...	...	1	1	3	1	1	2	2	4	3	4	4	1	0	2	1	2	3	...	...	...	...	...	...	...	35
University of Wisconsin	...	.	...	...	...	...	...	*	...	...	0	0	2	1	0	1	0	2	3	0	0	1	0	2	0	1	0	...	...	...	...	...	...	...	13

* This designates the year the course of instruction was established.

† Indicates when the school was founded.

The Efficiency of Built-Up Wooden Beams.

BY EDGAR KIDWELL,* HOUGHTON, MICH.

(Lake Superior Meeting, July, 1897.)

PREFACE.

To any one acquainted with the practical conditions surrounding the mining engineer and mine-manager, especially in this country, the presentation to the American Institute of Mining Engineers of a paper on the above subject will need no explanation. The members of this profession are continually called upon to design and erect, both above and underground, structures which involve the use of beams. Not to mention the various buildings connected with large mining operations, I need only refer, in support of this statement, to trestles, derricks, headworks, etc., on the surface, and to timbered stations and other spaces of large span in the mine-workings.

Notwithstanding the rapidity with which iron and steel are replacing wood in the construction of large buildings, there will always be cases in which the use of metal will be unadvisable. Especially will this be true in districts remote from large cities. Works of temporary character will continue to use timber. High freight-charges, the liability of delays in delivering iron, and the temporary character of many of the structures, will for many years to come make it advantageous for the mines west of the Mississippi and on Lake Superior to construct their buildings almost exclusively of wood.

The timber-resources of the country are decreasing so rapidly that already the engineer finds difficulty at times in obtaining beams of large size. This difficulty will steadily increase, and will force the constructor to resort more frequently to built-up beams, such as have long been used in Europe, owing to the difficulty of obtaining there such large sizes of timber as have

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hitherto been easily procurable in this country. Though located in one of the leading lumber-regions, the writer has occasionally found it necessary to use built-up beams, and other constructors have had a similar experience.

The experiments herein described were made in the mechanical laboratory of the Michigan College of Mines, with the object of obtaining data which I have found necessary for the efficient execution of work placed under my charge.

I take pleasure in acknowledging my obligations to my assistant, Mr. C. F. Moore, and to Mr. James Fisher, Instructor in Mathematics, both of whom rendered valuable aid by observing the deflections during the tests.

Further investigation will be necessary before all important points can be well worked out. The experiments required for this purpose will be made as soon as my other engagements permit, and results will be reported in a later paper.

Before designing the beams herein discussed all literature on the subject accessible to the writer was looked up, but it was found to be scanty in amount, and of little practical value. At the close of this paper is given a list of all references found.

So far as the writer knows, T. M. Clark is the only author who assigns any numerical value to the efficiency, or the ratio of the strength of a built-up to a solid beam of the same size.* Other writers are either silent on this point, or seem to think it necessary to warn the reader that a built beam is not *stronger* than a solid one. "A built beam cannot be stronger than a solid one (as some have supposed), when the material in the solid one is as good throughout as that in the built one" (De Volson Wood). "We obtain *nearly* the strength of a solid beam of the total depth" (Trautwine). "The strength of the former (*i.e.*, built beam) is to be relied upon, although it cannot be stronger than the corresponding solid one, if perfectly sound" (Wheeler, copied by Kidder).

None of these writers give other than rule-of-the-thumb methods for designing built beams, and even these rules are few. The extremely important consideration of the relative stiffness of solid and built beams is not even mentioned.

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* Since this was written the efficiency of several beams has been found in Smith's paper on "Yellow Pine Timber" (see page 818).

Beams designed according to these rules have not, when subjected to the full load for which they were calculated, in any case which has come under the writer's notice, borne out the statements just quoted. The chief defect in built beams is their great lack of stiffness. Some forms, even when lightly loaded, deflect to an extent which renders them useless in any building whose interior is to be finished.

The writer therefore decided to make such tests as would enable him to determine, with sufficient accuracy for all practical purposes, the relative strength and stiffness of built-up beams and solid beams. The pieces tested were not so large as the writer would have had them, but the mechanical facilities at his command, and the difficulty of obtaining sticks which would saw up into the desired check-pieces, prevented the use of larger sizes. It is believed, however, that the method of experimenting is such that the results here obtained are equally applicable to larger-sized beams.

After each set of tests are given the writer's conclusions concerning the form of beam tested. It will not be amiss, however, to state here in a condensed form a summary of the conclusions drawn from the whole work.

The results obtained seem to demonstrate that it is practically impossible to build a compound beam that will be as strong as a solid beam of equally good material. Indented beams are costly, wasteful of material, and extremely inefficient, and hence should be avoided. Clark's design is not based on correct principles, and hence gives a beam that lacks both efficiency and stiffness. Beams with inclined keys are costly to build, and are of low efficiency. The writer's design, based on the use of pipes, is a fairly cheap form, and excels Clark's in both efficiency and stiffness, but is inferior in both respects to keyed or joggled beams. The latter form appears to be in every respect superior to any other. Blocking apart the component pieces of a keyed beam does not, contrary to the generally accepted view, increase the efficiency to any appreciable extent, but does increase the stiffness under moderate loads.

The greatest defect in all built-beams is the tendency of the keys or other fastenings to rotate in their slots, and unless such rotation is prevented the component parts of the beam slip, and both the efficiency and stiffness are impaired. All washers for

bolts should be extra large and heavy. The ordinary cut washers are entirely too small and light. The writer advises cast washers whose diameter is four to five times that of the bolt, and thickness equal to not less than three-quarters of diameter of bolt. Numerous small keys closely spaced give better results than fewer larger keys, since the rotation can be more easily prevented, and the beam itself is not weakened by large slots. It is impossible to make an efficient beam by using bolts alone as fastenings, as bolts easily bend, and the pieces slip.

The resistance of soft wood to end-pressure is so much greater than that of hard wood to side-pressure, that hard-wood keys, even in beams of white pine or other soft wood, become indented, and allow the component parts to slip, thereby increasing deflection of the beam. Oak keys certainly cannot, especially if the span of the beam is short, give satisfactory results in beams built of oak, the yellow pines, or other hard wood, and either a much stronger wood for keys must be got or iron keys must be used, if the construction is to be in reality what it often is only in appearance. The breadth of a key should never be less than twice the thickness for permanent work, and two and a half times the thickness would be much better.

In the center of the beam a length equal to one-quarter of the span should be left free of keys or fastenings of any kind, unless, perhaps, a few lag-screws passing through the compression-member, but not more than half-way through the tension-member.

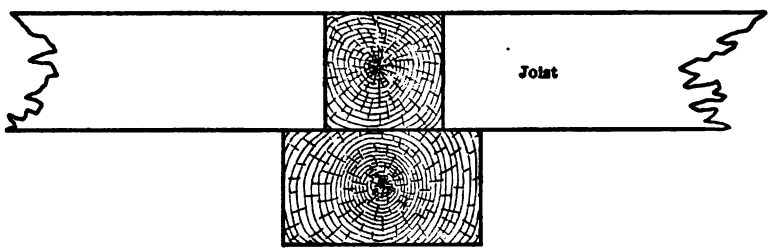
The yielding of the wood around bolts or pipes, against keys, etc., the penetration of end-wood into end-wood in indented beams, and the slight rotation of keys in their slots, all conspire to produce in a built beam a set of conditions different from those existing in a solid piece. The effect of this is to render the deflection of a built beam greater than that of a solid one. Conditions existing in a solid beam are most nearly reproduced in a built one when a number of small, broad, iron keys are used. Since these do not indent, the slip is less; hence, stiffness is greater, and by proper proportioning such a built beam can be made practically as stiff as a solid one, as shown in the cases of beams Nos. 28 and 29 A.

The keys should be so located that the faces receiving a

thrust are perpendicular to the joint between the sticks. When inclined keys are used, as in Rankine's and Brunel's beams, the component pieces slide up the inclined faces of the keys, pry apart the joint, and the benefit of friction between the faces of the sticks is lost. Neither great strength nor stiffness can be secured by any form of key not provided with wedges which can be firmly driven up. Any other construction, regardless of the care used in fitting keys to their slots, leaves more or less slack, which is taken up as the load is applied, so that the component parts of the beam slip.

The wide range of efficiencies, even in beams of precisely the same design, shows the impossibility of getting equally close work in every case. In view of the fact that the writer personally built every beam with great care, and that the same

FIG. 1.



OBJECTIONABLE FORM OF BUILT BEAM.

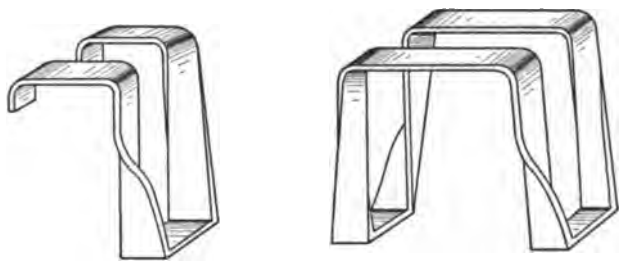
grade of work can hardly be expected when ordinary workmen are employed, the writer would, in case of any permanent structure, hesitate to count on over 75 per cent. efficiency in beams with joggle-keys,* or 65 per cent. in any of the other patterns.

It remains only to call attention to certain precautions which, as the writer knows from experience, are seldom observed. Built beams are often made as in Fig. 1, the joists resting on the lower piece. Sometimes the joists rest on a ledge-piece spiked on the lower half of the compound beam. The writer considers these methods very objectionable. In nearly all of the tests the tension-side failed first. It is evident that when the pressure of the joists comes on the tension-piece in the built beam, the piece tends to act independently, like a continuous

* Eighty per cent. for beams like Nos. 28, 29 A and 29 B.

beam whose spans are the distances between bolts. If the bolts are in the slightest degree slack, the tension-piece drops away from the upper piece in compression; and hence the tendency of the latter to act independently of the lower piece is increased, while the space made between the beams greatly facili-

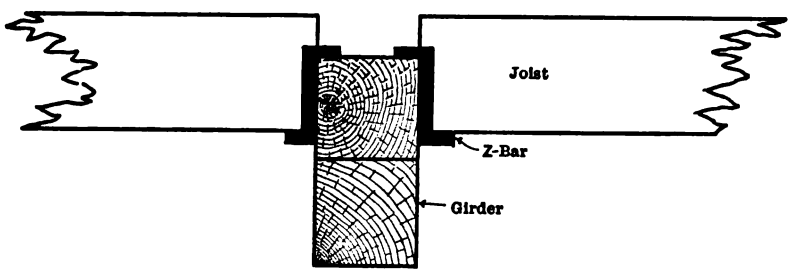
FIG. 2.



BRIDLE IRONS.

tates bending of the bolts and rotation of the keys. The weight of the floor ought to go on top of the beam, and if the joists cannot be made to bear directly on the built beam, they should be swung in bridle-irons (Fig. 2), or in some of the patent

FIG. 3.



SUGGESTED USE OF Z-BARS.

beam-hangers. The writer thinks that light Z-bars, as in Fig. 3, would give excellent results, but has not had an opportunity to try them.

PART I.

I. CONDITION AND METHOD OF EXPERIMENTS.

Kind and Grade of Lumber Used.—All the lumber was purchased of the Sturgeon River Lumber Co., Chassell, Mich.

The Norway pine used in tests described in Part II. was very

dry, and all dressed to size at the mill. The pieces were cut out of large beams which had been piled on the company's dock for a couple of years. They were free from large knots, shakes, etc., but a few of them had some sap-wood, and knots ranging from pin size up to one half-inch in diameter. The best of the large beams were selected, and the small ones then sawed out, without reference to how the grain ran in the pieces. All of this Norway pine was of a better grade than would be used in an actual structure, unless special care were taken in selecting the pieces.

Beams numbered 0 to 50 in Part I. were constructed of white pine, except where otherwise noted in the report. That the efficiencies determined from the tests might be unaffected by variation in strength of the pieces used in the beam, it was considered necessary to use No. 1 clear white pine, each piece specially selected so as to obtain a uniform and straight grain from end to end. The attempt was made to secure beams so cut that one side would be tangent to one of the annual rings, in the center of the beam, but it was not possible to obtain a number of such pieces sufficient for all the tests.

The white pine, except in beams Nos. 8, 9, 25, 26, 27, and 36 to 50, was sawed in June, stacked to dry till August, then dressed and delivered at the College of Mines, where it was piled so as to be exposed to ample draught of air. The other white pine was sawed in August, and immediately dressed and delivered. It had the benefit of five weeks' drying by steam before it was used. No moisture-determinations were made, as the use of the check-pieces rendered such determinations unnecessary.

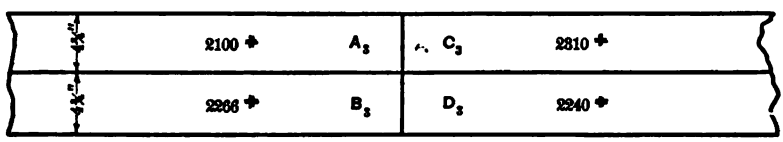
Building Up the Beams.—It seemed to the writer that if the tests were to be of any value the workmanship of the beams must be of the same grade throughout, and of first-class quality. He found it useless to expect an ordinary carpenter to carry out instructions with sufficient care, or understand the purpose of the tests; hence it became necessary either to abandon the work or build the beams himself. He therefore personally built and tested every beam herein mentioned.

All the white pine was dressed to exact thickness at the mill. After the lumber was delivered the pieces for testing were carefully laid out to gauge, then sawed square to size, so that no planing was needed to even up the widths.

Order of the Experiments.—In the following report the beams are numbered consecutively for convenience, but those numbers have no reference to the order in which the tests were made. The first work was done on the Norway-pine beams described in Part II. The intention was to test a number of single sticks, determine their average strength, and then use that average as a basis for computing the efficiency of the built beams. As the work progressed, it became evident that the variation in the strength of the single pieces was entirely too great to render the average a safe basis for figuring the efficiency of the built beams. It was, however, considered worth while to continue these tests, for reasons that are given in Part II.

It is clear that the essential point is to determine the efficiency of the joint between the pieces composing the beam, entirely independent of the usual variations in the strength of separate pieces of timber of the same kind. It appears to the

FIG. 4.



SHOWING METHOD OF CUTTING OUT CHECK PIECES.

writer that the only practicable method of doing this, or anything like it, is to provide a separate check on each built-up beam by testing pieces cut from the same plank from which were taken the pieces used to build up the beam. Further, these check-pieces must have the same grain, etc., as those used in the built beam. This plan was accordingly followed in beams, Nos. 0 to 50.

Most of the white pine was in pieces 9.625* inches wide, 2.75 inches thick, and 12 feet long. The best method of procedure seemed to be to saw this plank into four pieces, each 4.75 inches wide, 2.75 thick, and 6 feet long, then use for the built beam two pieces diagonally situated, reserving the other two for testing singly, to obtain a basis on which to figure the efficiency of the built beam. *E.g.*, the plank would be sawed as in Fig. 4, then A and D would be used for the built beam, while B and C would be tested singly.

* See foot-note, page 747.

Before deciding on this plan, a wide board was cut as in Fig. 5, and each piece was tested with the following results.

In all the tests the loads were applied at the center of the span.

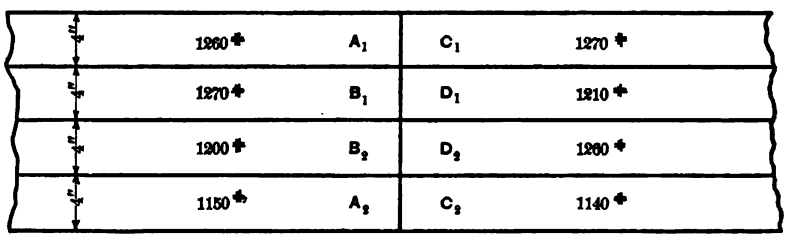
$A_1 = 1260$	$B_1 = 1270$	$A_2 = 1150$	$B_2 = 1200$
$D_1 = 1210$	$C_1 = 1270$	$D_2 = 1260$	$C_2 = 1140$
Sum, 2470	Sum, 2540	Sum, 2410	Sum, 2340

A board as in Fig. 4 was then cut up and tested as follows:

$A_3 = 2100$	$B_3 = 2266$
$D_3 = 2240$	$C_3 = 2310$
Sum, 4340	Sum, 4576

In view of these results, the foregoing plan was considered satisfactory, and was followed in all tests included in Part I.

FIG. 5.



It must be remembered, however, that it is impossible to get sticks of a quality so uniform throughout as to justify the belief that the method of using check-beams, as here done, gives absolutely correct results. That the results are more accurate than anything hitherto obtained, the writer is fully convinced; yet inspection of the differences between the ultimate loads of the check-beams themselves demonstrates that mathematical accuracy is not, and probably cannot be, obtained in work of this nature. The results of beam No. 24 show this.

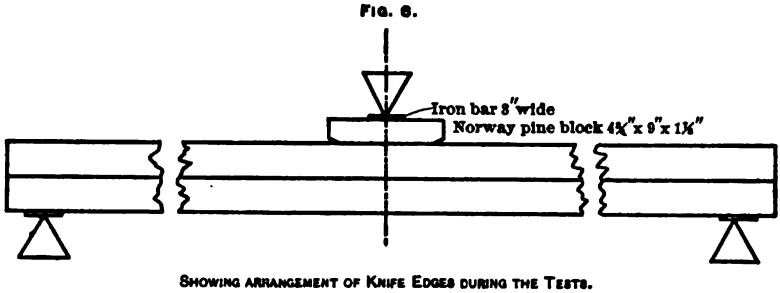
"Both Prof. Lanza's tests on wooden beams and those conducted by the author show that beams having a shorter length than 20 times the depth under a uniformly distributed load, or less than 10 times the depth under concentrated loads, should be dimensioned for shearing lengthwise."* Further, it is

* *Materials of Construction*, J. B. Johnson, First Edition, p. 681.

mathematically shown in Part III. that the efficiency of a built beam is necessarily greater with large than with small ratios of span to depth. It was therefore considered advisable in these tests to make the ratio near the critical value of 10, hence the beams in Part I. are mostly proportioned to a ratio of 12, while a ratio of 9.6 is employed for the majority of beams in Part II.

Conducting the Tests.—All the testing was done on a Riehlé 100,000-pound testing-machine, the beam of which is graduated to read to 10 pounds. To adapt the machine to testing pieces up to 9-foot span, the platform was extended by means of three 6-inch steel I-beams, 9 feet 6 inches long. The knife-edges accompanying the machine were then elevated the proper amount by placing them on hard pine blocks, the bearing-area of the cast-iron block carrying the knife-edge being sufficiently

FIG. 6.



SHOWING ARRANGEMENT OF KNIFE EDGES DURING THE TESTS.

large to prevent any appreciable indentation of the hard pine. Iron plates 3 inches wide and 0.5-inch thick were placed between knife-edges and the beam, as shown in Fig. 6. The pressure on the beam was applied through a Norway-pine block, whose under-edges on each end were beveled for a distance of 1 inch, as shown in the figure.

Measuring the Deflections.—No means of measuring the deflections with great accuracy were available. It is believed, however, that the results obtained by the following method are accurate enough for all practical purposes, as is clearly shown by the curves in the deflection-diagrams.

A steel nail was driven into one side of each beam, in the neutral plane and center of the span. This nail was then carefully sharpened, and the point was made to move over a steel scale graduated to sixteenths of an inch. An assistant read off

the deflections while the writer noted the readings of the beam, and attended to the operation of the machine.

It was considered unnecessary to measure the deflections beyond a point where the load on the beam was over twice what would be used in practice, even with a small factor of safety.

Reporting the Tests.—Owing to the dearth of information on the subject of this paper, it has been considered advisable to give a full report of each test, so that the reader will be put in full possession of all the facts necessary to enable him to draw his own conclusions, which may not always coincide with those of the writer. The drawings show all details of construction of the beams. The superior value of graphical illustrations is so generally conceded, that it has been considered best to omit the usual tables, and to plot deflection-diagrams which enable the behavior of the beams to be understood at a glance. The latter is a most important point, since the question of stiffness in a beam is really more important than that of high efficiency. A beam highly efficient, but lacking in rigidity, would be worthless for most purposes of construction.

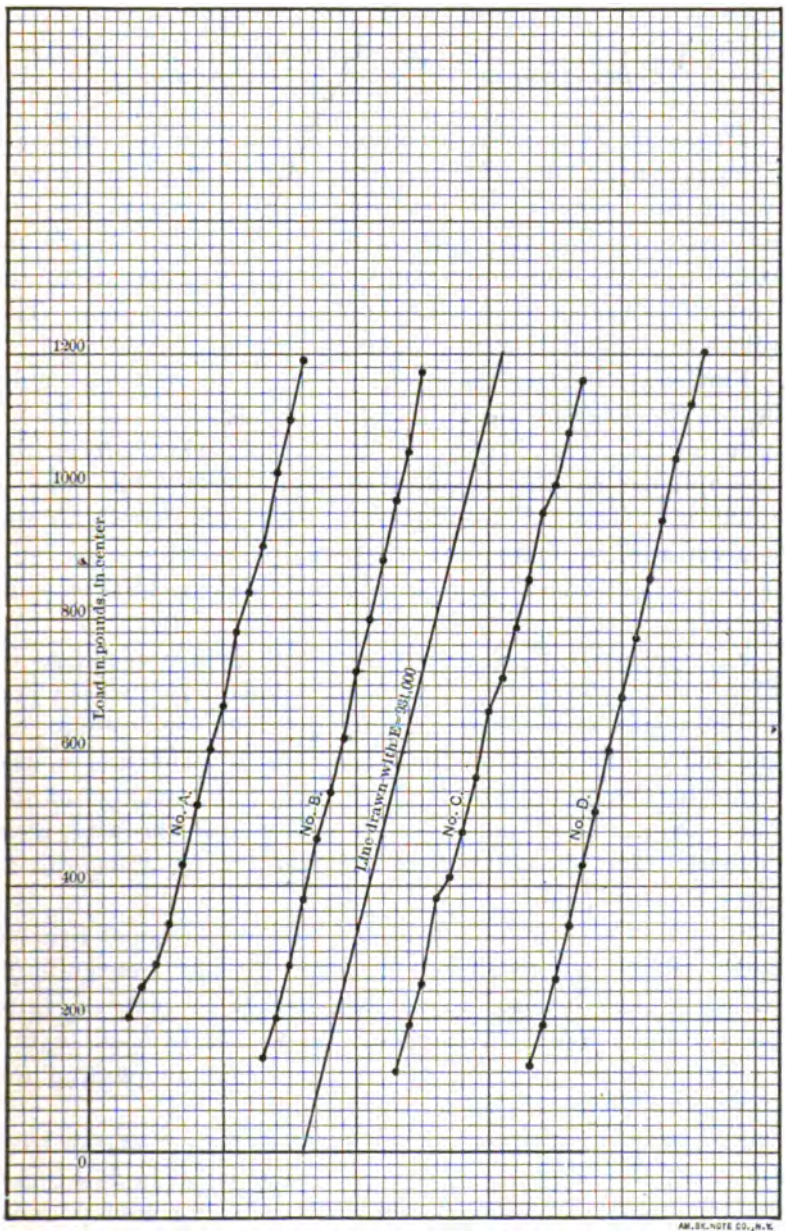
Computing the Efficiency.—All of the white pine built-beams and check-pieces were weighed previous to testing. These weights could have been neglected without material error, but it was thought best to take them into account. No correction was made for the length of beam overhanging knife-edges, but, to offset this, the weight of the Norway-pine block used for distributing the pressure applied by the drawhead of the machine was neglected. The efficiency was computed as follows: The sum of the ultimate strengths of the two check-pieces was doubled, and to this was added the weight of these two pieces. This result gives four times the average strength of each check-piece, which amount would be the strength of a solid beam double the height of the check-piece, and of exactly the same quality. This value was taken as unity, and the ultimate strength of the built beam, corrected for weight, was referred to it to find the efficiency.

Example.—In beam No. 1,

Check-piece A, weight 17 pounds, broke at 2700 pounds.
" " D, weight 18 " " 2600 "
Sum, 35 " Sum, 5300 "

$2 \times 5300 + 35 = 10635.$ = probable strength of solid beam.

DIAGRAM 1.



DEFLECTION TESTS ON SINGLE STICKS.

Each horizontal space=1/32 inch deflection.

Built beam weighed 42 pounds. Broke at 7500 pounds.
Correct for weight, 21 "
7521 "

$$\text{Hence, efficiency} = \frac{100 \times 7521}{10635} = 70.7 \text{ per cent.}$$

Plotting the Deflections.—Diagrams 1 to 14 show the curves or "characteristics" of all beams whose deflection was measured, except two. Some of the beams were one-quarter to one-half inch narrower than the others, and in such cases the loads obtained from the tests were multiplied by the proper factor to convert the beams to the standard width of their class. The values thus obtained were used in plotting the curves.

Further, to indicate the relative stiffness of solid and built beams, the diagrams for Part I. show characteristics for a solid beam of the same size as the built ones, calculated on the assumption that the modulus of elasticity of white pine is 1,000,000. That this assumption is practically correct for the material used in these tests is shown by the curves for built beams Nos. 28 and 29 A, as well as the following tests on solid beams:

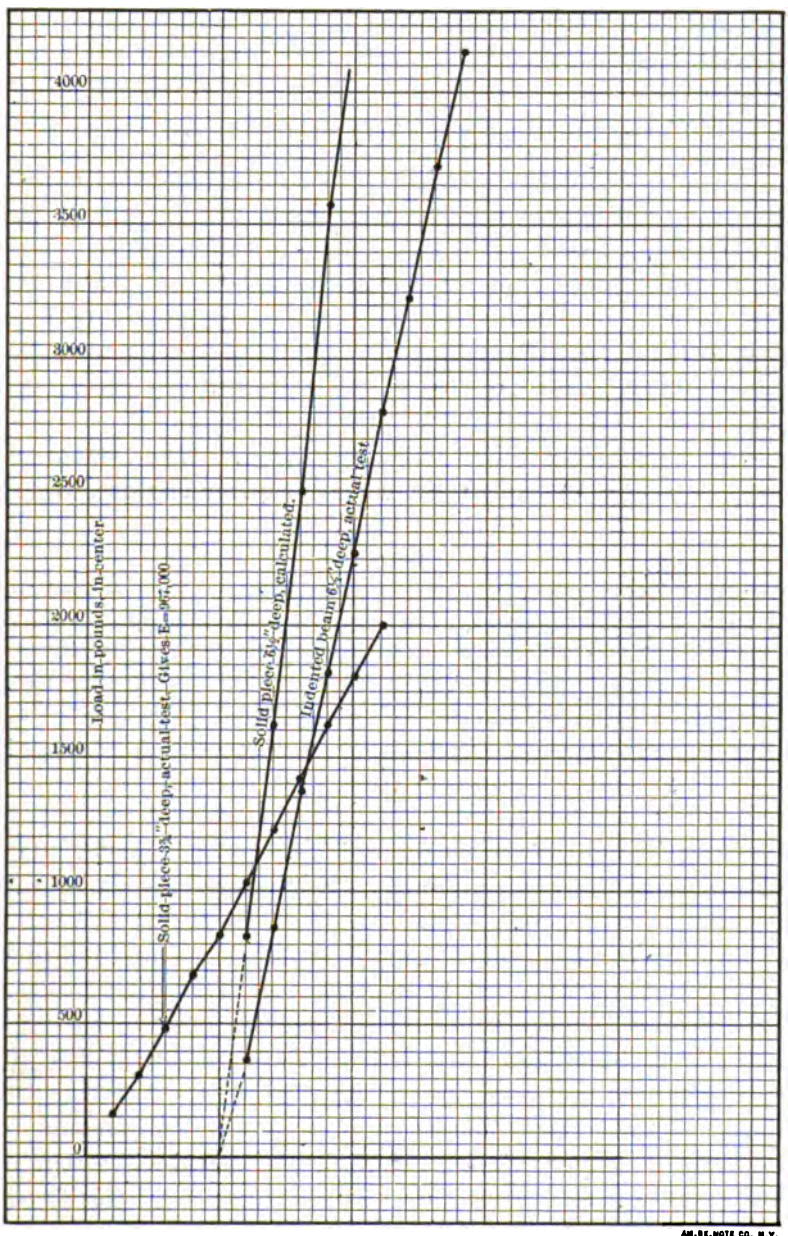
Four pieces, A, B, C, and D, were cut from a plank as in Fig. 4, each piece being 2.75 inches deep, 4.75 inches wide, and 66 inches span. These were subjected to deflection-tests, and the results are plotted in Diagram 1. The line in the center has been drawn to represent about the average of the four curves. This line gives the modulus of elasticity, E = 931,000. Test on check-piece for beam No. 0 (Diagram 2) gives E = 967,000.

Since the Association of Railway Superintendents of Bridges and Buildings has recommended (Fifth Annual Convention)* that 1,000,000 be used as the modulus of elasticity of white pine, and the results of the tests are so nearly equal to this value, it has been thought best to use it here in plotting the theoretical characteristics for solid beams.

No correction has been made for the deflection of the steel I-beams. The moment of inertia of each beam is 28.4. Assuming a span of 66 inches, and a center-load of 4000 pounds, the girder composed of three I-beams will deflect only 0.009-

* *Proceedings* for 1895.

DIAGRAM 2.



INDENTED BEAM

Each horizontal space = 1/32 inch deflection.

inch. The actual deflection was less than this, since, owing to the width of the machine-platform, the weight was distributed

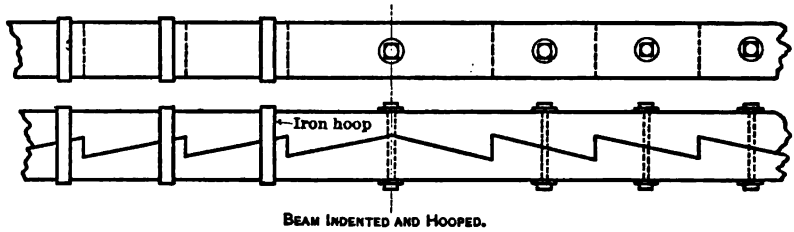
over a distance of 27 inches. It is therefore evident that the deflection of the steel beams was too trifling to be taken into account.

II. Forms of Beam not Considered Worth Testing.

As the experiments proceeded it became evident that certain forms of built beam more or less recommended by some writers are devoid of merit. It is well known to engineers that many of the complicated joints used in carpentry, even at the present day, are of no value, and are merely the survival of methods employed in ancient times, when bolts and screws were unknown or too expensive to use, while timber was cheap, and joints were made mostly with hard-wood tree-nails.

Fig. 7 represents a form much advocated. When straps are used, as shown on the left, the beam is tapered slightly, so that

Fig. 7.



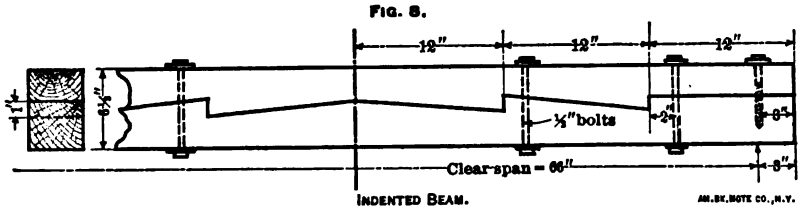
Beam Indented and Hooped.

the straps can be driven up tight, like hoops on a barrel. The objections to this beam are many. If straps are used, they will be very costly if properly made, and very inefficient unless the beam is accessible so that the straps can be driven up from time to time, until the wood is perfectly seasoned. The writer does not believe that it is possible for the average carpenter to lay out a beam of this kind and work with such accuracy that each indent will bear evenly on the one adjacent to it. Nor can this defect be remedied by making a driving fit. The amount of work necessary for forming the pieces is so excessive that in the opinion of the writer the only consideration to justify the use of this form at all would be an efficiency superior to that of any other form. It is evident at a glance that such efficiency cannot be got; but in order to remove any doubt, the following test was made:

Indented Beam No. 0 A.—Two pieces of white pine 3.75* inches deep, and 4.5 inches wide, notched down one inch, giving built beam 6.5 inches deep. Pieces secured by four 0.5-inch bolts and two 0.5-inch lag-screws, 6 inches long. (See Fig. 8.) Span, 66 inches. Weight of beam, 42 pounds. Check-beam, 3.75 inches deep, 4.5 inches wide, and weighed 20 pounds. Pieces cut from one plank, as in Fig. 22. Both beams were tested for deflection. Check-beam broke at 2960 pounds, built beam at 6180 pounds. The curve in Diagram 2 shows that the deflection of the built beam was just double that of a solid stick of the same size.

Since the strength of a beam varies as the square of its height, the ultimate load of a solid beam 6.5 inches deep would be $(2960 + 10) \times \left(\frac{6.5}{3.75}\right)^2 = 8923$ pounds, and the efficiency of the built beam $= \frac{100\ (6180 + 21)}{8923} = 69.5$ per cent.

FIG. 8.



Since the indents cut off 1 inch from the depth which might have been secured by using other methods, the efficiency, referred to 7.5 inches depth, will be $\frac{100 \times (6180 + 21)}{4 \times (2960 + 10)} = 53$ per cent.

Assuming that these two pieces had been built up by Clark's method, which is about the poorest of those to be described, the ultimate load would have been, on a basis of 71 per cent. efficiency, 8434 pounds, against 6180 as shown in the test.

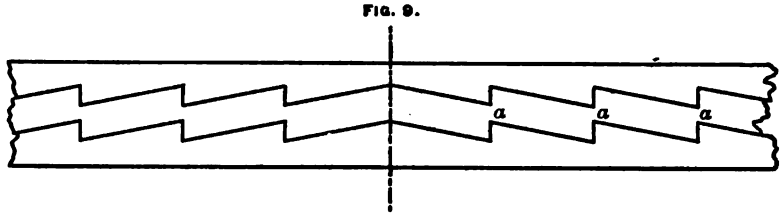
To determine the deflection curve for a solid piece 6.5 inches deep, the loads for each deflection-reading in test of the check-beam were multiplied by $\left(\frac{6.5}{3.75}\right)^3$, and the resulting curve is plotted in Diagram 2.

* All measurements of test-pieces were to even sixteenths of an inch. For the convenience of the printer, these are given in decimal form hereafter.

These results show clearly that this form of beam is to be avoided; hence, no further tests were made on it or its modifications. The test developed the fact that where end-wood bears on end-wood the fibers easily penetrate one another like a series of wedges, and these allow the component pieces to slide, thereby increasing the deflection and cutting down the efficiency.

The writer believes that this form would give better results

FIG. 9.

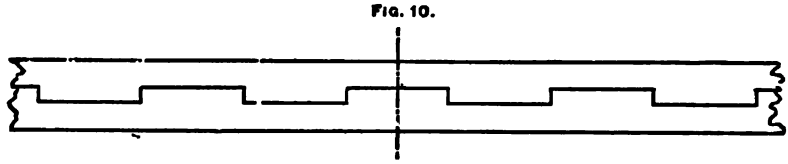


INDENTED BEAM.

were driving keys, or even slips of tin, put between the faces of indents to prevent end-wood from coming into contact.

Fig. 9, from Mahan's *Civil Engineering*, shows a method of applying to a beam three pieces deep the idea embodied in the beam just discussed. This is clearly a worse design than the other, as the waste of material and the labor of building up are very great; the sag will be excessive, and fracture will soon

FIG. 10.



INDENTED BEAM.

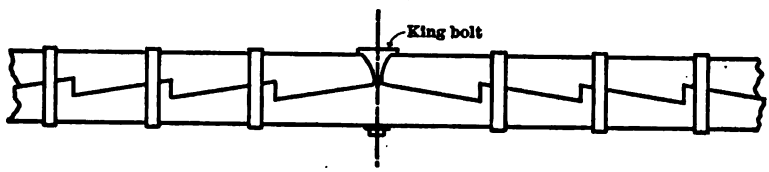
occur at *a*. It was considered useless to test this form, or the ones immediately to be described.

Fig. 10 represents the same idea applied in a somewhat different manner, and has no merit over the preceding form, except that it might, perhaps, be cheaper to build. It is clearly a waste of good material.

Fig. 11, from Tredgold's *Carpentry*, shows the same idea, but the king-bolt with wedge-shaped head has been added, in order to jam the faces of the indents and prevent the beam from sag-

ging. This expedient evidently does little good, since it in no way makes up for the failure to space the indents at exactly equal distances, and merely creates in the bottom-piece an initial tension which will cause the beam to fail sooner than otherwise. It also requires a hole through the tension-side of the beam at the point where the bending-moment is greatest, which is a serious defect.

FIG. 11.

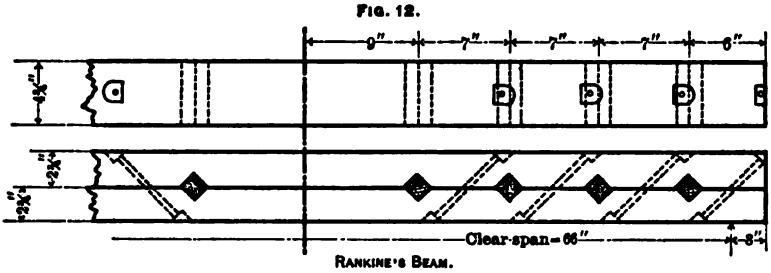


TREDGOLD'S BEAM.

Fig. 12 represents a beam advocated by no less an authority than Rankine. He says:

"Considering that the stress at the neutral surface is equivalent to thrust in a direction sloping at 45°, combined with tension in a direction sloping at 45° the opposite way, it would seem that the best position for the keys would be that shown in (our) figure, their fibres being made to slope in the direction of the thrust, and the bolts being made to slope in the direction of the tension. This, however, so far as I know, has never yet been tried."*

FIG. 12.



RANKINE'S BEAM.

KEYS OF WHITE OAK 1¼ INCH SQUARE, HALVES TAPERED AND DRIVEN UP.

The writer does not believe that this reasoning is perfectly sound. The stresses mentioned refer entirely to the case of a solid beam; nor could any amount of bolting, etc., change them in any way, or add any strength to a solid beam. The problem in the case of a built beam is how most effectually to prevent motion along the joint between the component sticks; and it is self-evident that this can best be done when the pres-

* *Civil Engineering*, page 463.

sure on the keys, bolts, or other means used, is resisted by surfaces at right angles to the direction of motion. Since the tension-side must elongate, it is clear that as soon as Rankine's beam is loaded the bolts will tend to become vertical, fail to hold the pieces firmly together, and, in consequence, permit the component pieces to slide up on the inclined faces of the keys. The result will be failure of the beam under moderate load.

This construction is also expensive. The bolts would have to be provided with cast-iron seats under washers, or else notches large enough to hold the washers would have to be cut in the timber, thereby causing a serious loss of strength, especially on the tension-side.

The correctness of this view is evidenced by the following:

III. Tests on Rankine's Beams.

No. 0 B. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide. Bolted and keyed as in Fig. 12. Span, 66 inches; weight of beam, 32 pounds. Broke at 8570 pounds. Check-pieces: B weighed 13 pounds, broke at 2870 pounds; C weighed 13 pounds, broke at 3440 pounds. Efficiency of built beam, 67.9 per cent.

No. 0 C. Same as 0 B. Built beam weighed 35 pounds, broke at 5900 pounds. Check-pieces: B weighed 14 pounds, broke at 2930 pounds; C weighed 14 pounds, broke at 2080 pounds. Efficiency of built beam, 58.8 per cent. Average efficiency from the two tests, 63.3 per cent.

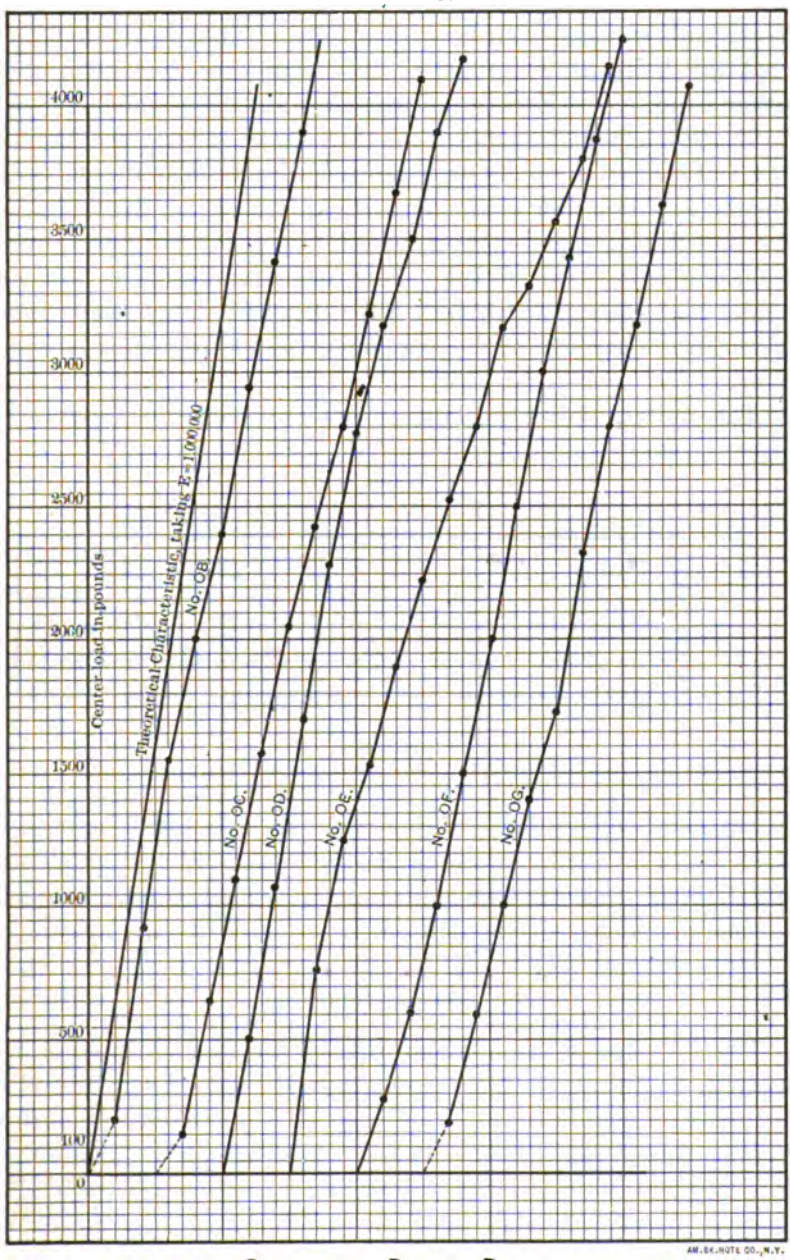
Owing to the peculiar construction of this beam it was thought desirable to test two of them turned upside down.

No. 0 D. Same as 0 B, except that the sticks were 4.5 inches wide, and the beam was inverted. Weight of built beam, 36 pounds; broke at 5700 pounds. Check-pieces: B weighed 15 pounds, broke at 3500 pounds; C weighed 14 pounds, broke at 3200 pounds. Efficiency of built beam, 42.6 per cent.

No. 0 E. Same as 0 D. Beam weighed 32 pounds, broke at 4750 pounds. Check-pieces: B weighed 13 pounds, broke at 2670 pounds; C weighed 14 pounds, broke at 2880 pounds. Efficiency of built beam, 42.8 per cent. Average of 0 D and 0 E, 42.7 per cent.

Diagram 3 shows that the first two beams started very well,

DIAGRAM 3.



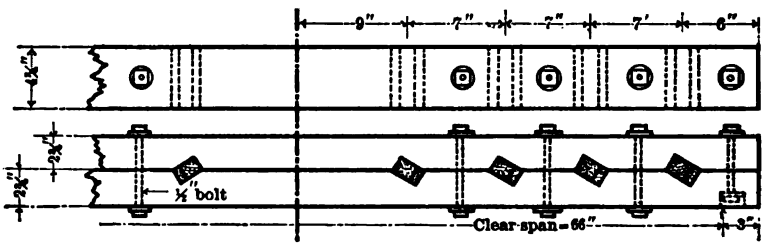
RANKINE'S AND BRUNEL'S BEAMS.
Each horizontal space = 1/16 inch deflection.

but as the load increased the deflection became marked. This was wholly due to the straightening out of the bolts. The best result was got from beam No. 0 D, which was inverted, so that the bolts drew tighter as the loads increased. On the other hand, No. 0 E gave the worst results.

The low efficiencies speak for themselves. They are due to the fact that as the bolts drew more and more near the perpendicular position the component sticks slid up on the inclined faces of the keys, and sprung the joint open three-eighths of an inch before the beam failed. This caused excessive deflection under high loads, and prevented the usual frictional resistances between the surfaces of the component sticks from coming into play.

The writer, therefore, fails to see any merit in Rankine's

FIG. 13.



BRUNEL'S BEAM.
KEYS OF WHITE OAK, 1 X 2 INCHES, HALVES TAPERED AND DRIVEN UP.

beam, and believes that, judging from No. 0 D, much better results could be got by inclining the bolts the other way, about 10 degrees from the perpendicular, provided proper shaped washers of ample size were used.

IV. BRUNEL'S BEAM.

Fig. 13 shows a beam much used by the distinguished engineer, I. K. Brunel.* The model tested differs from Brunel's actual construction to the extent that all keys on one side of the center of span are turned the same way, while Brunel made the end-key face in a direction opposite to the others next to it. A key so placed seems to do little or no good, since the com-

* *Cf.* "Transverse Strength of Large Beams of Yellow Pine Timber," H. D. Smith, *Proceedings of the Institution of Civil Engineers*, vol. cxxviii., p. 323.

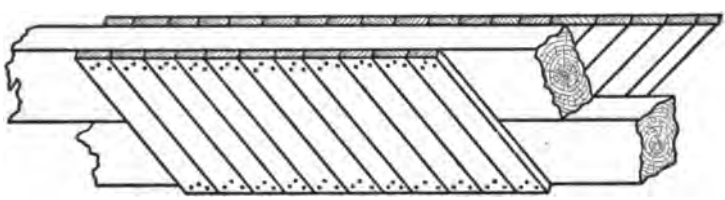
ponent sticks slide over it; hence, that arrangement was not used in the tests.

No. 0 F. Two pieces of white pine, 2.75 inches thick and 4.75 inches wide, keyed and bolted as in Fig. 13. Span, 66 inches. Built beam weighed 33 pounds, broke at 9780 pounds. Check-pieces: B weighed 13 pounds, broke at 2690 pounds; C weighed 14 pounds, broke at 2860 pounds. Efficiency of beam, 88.0 per cent.

No. 0 G. Exactly the same as 0 F. Beam weighed 34 pounds, broke at 8660 pounds. Check-pieces: B weighed 15 pounds, broke at 2800 pounds; C weighed 14 pounds, broke at 3120 pounds. Efficiency of beam, 73.1 per cent. Average efficiency from the two tests, 80.6 per cent.

Diagram 3 shows that this construction gives a good stiff beam. The efficiency is not, however, as high as that of the

FIG. 14.



ISOMETRIC VIEW OF CLARK'S BEAM.

plain joggled beam, for the same reason as in Rankine's construction—the component sticks ride up on the faces of the keys and force open the joint. This could be prevented only by a number of bolts and size of washers that would be prohibitive. The writer also found that much more time is necessary to make a good fit between keys and their slots than is necessary with the plain joggle-keys, and therefore does not recommend any form of beam with inclined keys.

V. CLARK'S BEAM.

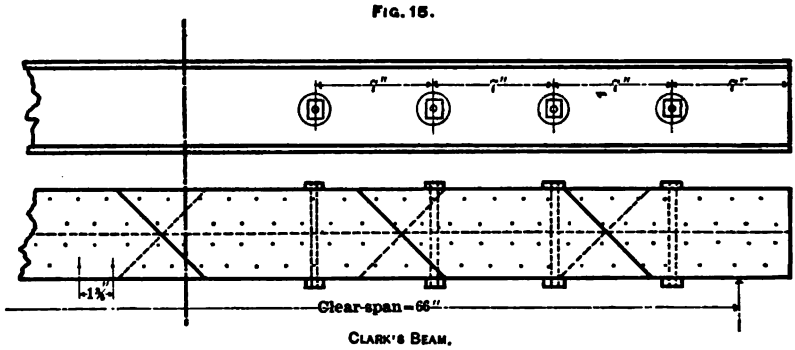
The construction of this beam is shown in Fig. 14. Clark says that the idea was suggested by the design of certain bridge-trusses, and adds:

"The action of the oblique struts in the bridge can be applied to the compound beam in the simplest possible way by nailing oblique pieces of board firmly to the timbers, reversing them on opposite sides, in imitation of the

reverse struts in the bridge-truss. If these boards are 1.25 inches thick, their united strength will be greater than that of either of the beams;* and if well nailed, so as not to spring, the lower beam will be torn apart before they will yield. The nails, if long enough to penetrate three inches or more into the girders, can only give way by shearing, which would require a force greater than would be necessary to break a solid beam. The strength of a girder built up in this way will not be affected by shrinkage, which soon causes more or less deflection in those indented or keyed together; for its resistance is maintained by the board struts, which act only in the direction of their length; and this remains invariable, whatever may be the lateral shrinkage."†

To test this beam, Clark constructed models one-eighth of actual size, making the component parts 1.5 inches deep and 1 inch wide. On the same scale the model would have been 30 inches long, but in order to decrease the weight necessary to break it, he made the length 10 feet. The model showed a strength equal to 95 per cent. of that of an equivalent solid

FIG. 15.



CLARK'S BEAM.

beam, and Clark assumes this value to hold good for the actual girder composed of two 3- by 12-inch pieces of 20 feet span.

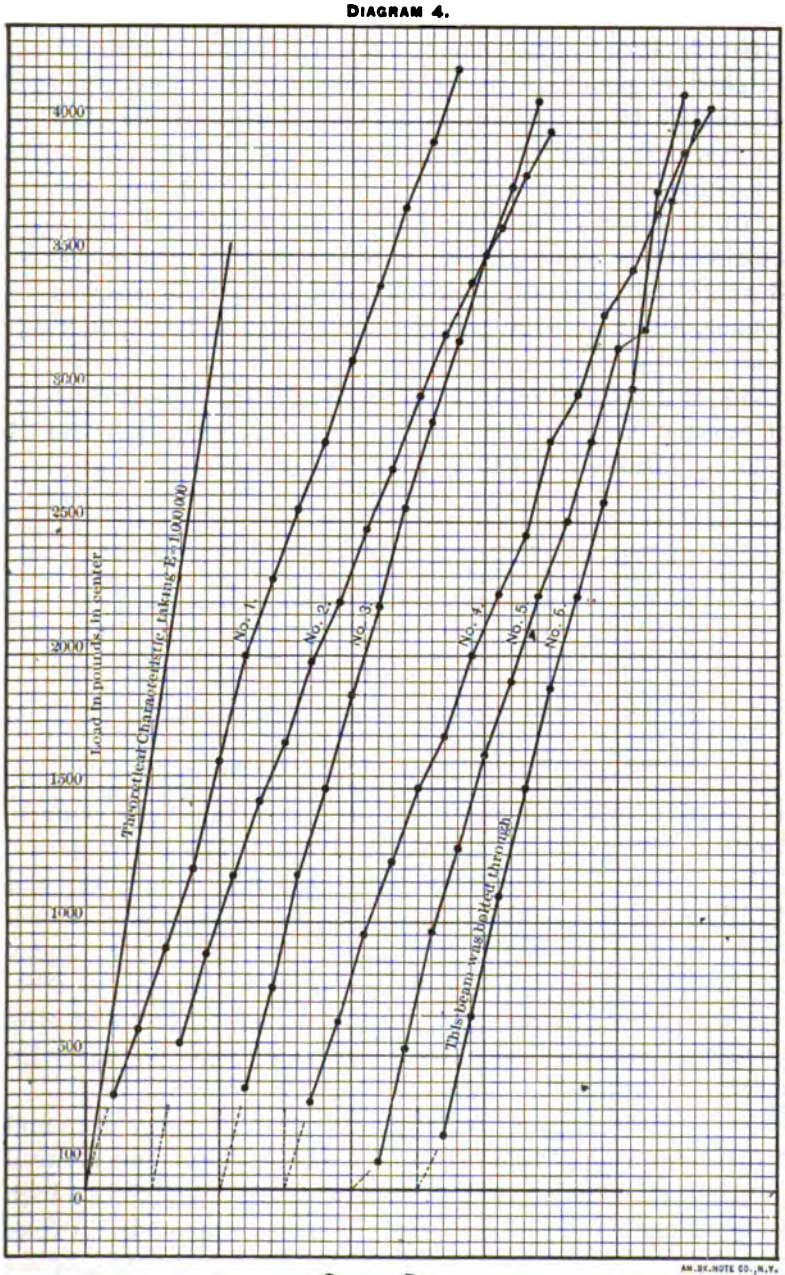
In the tests to be described, white pine was used for side-struts in some of the beams, and selected Norway pine in the others. Diagram 4 shows deflections.

No. 1. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide. Side-struts of white pine 0.5-inch thick, secured by 2-inch steel wire nails. No bolts used. Span, 66 inches. (See Fig. 15.) Weight of beam, 42 pounds. Broke at 7500 pounds. Check-pieces: A weighed 17 pounds, broke

* In the case under discussion, two 8- by 12-inch pieces of Georgia pine were to be built into a compound beam.

† *Building Superintendence*, page 91.

DIAGRAM 4.



CLARK'S BEAM.

Each horizontal space = 1/12 inch deflection.

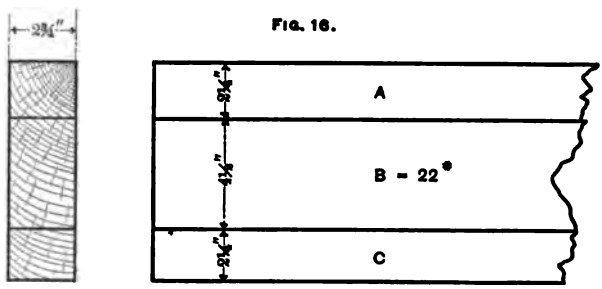
at 2700 pounds; D weighed 18 pounds, broke at 2600 pounds. Efficiency of built beam, 70.7 per cent.

No. 2. Exactly the same as No. 1, except that the beam weighed 38 pounds. Broke at 5200 pounds. Check-pieces: A weighed 16 pounds, broke at 1880 pounds; D weighed 16 pounds, broke at 1980 pounds. Efficiency of built beam, 67.3 per cent.

No. 3. Exactly the same as No. 1, except that the beam weighed 42 pounds. Broke at 7800 pounds. Check-pieces: B weighed 18 pounds, broke at 2400 pounds; C weighed 18 pounds, broke at 3500 pounds. Efficiency of beam, 72.2 per cent.

No. 4. Exactly similar to No. 1, except that side-struts were of selected Norway pine 0.5-inch thick, and beam weighed 37 pounds. Broke at 6180 pounds. Check-pieces: A weighed

FIG. 16.



15 pounds, broke at 1830 pounds; D weighed 17 pounds, broke at 2150 pounds. Efficiency of beam, 77.6 per cent.

No. 5. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide. Side-struts of selected Norway pine 1 inch thick, secured by 3-inch wire nails. No bolts used. Span, 66 inches. Weight of beam, 50 pounds. Broke at 6980 pounds. Check-pieces: B weighed 16 pounds, broke at 2150 pounds; C weighed 17 pounds, broke at 2640 pounds. Efficiency of beam, 72.8 per cent.

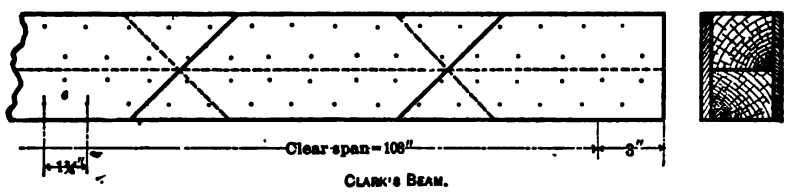
No. 6. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide. Side-struts of 0.5-inch white pine, secured by 2-inch wire nails. Pieces further secured by eight 0.5-inch bolts. Span, 66 inches. Weight of beam, 45 pounds. Broke at 8400 pounds. Check-pieces: B weighed 16 pounds, broke at 3000 pounds; C weighed 16 pounds, broke at 3190 pounds. Efficiency of beam, 67.9 per cent.

No. 7. Two pieces of white pine, each 2.25 inches deep and 2.75 inches wide. Side-struts of 0.5-inch white pine, secured by 2-inch wire nails. Span, 108 inches. (See Fig. 17.) Weight of beam, 34 pounds; broke at 1680 pounds. Check-piece was 2.75 inches wide and 4.5 inches deep. Weight, 22 pounds; broke at 2600 pounds. Efficiency of built beam, 65 per cent.

Fig. 16 shows the method of cutting out the various pieces.

No. 8. Two pieces of white pine, 2.25 inches deep and 2.375 inches wide. Side-struts of 0.5-inch white pine secured by 2-inch wire nails. Span, 108 inches. (See Fig. 17.) Check-piece, 4.5 inches deep and 2.375 inches wide. Built beam, weight, 28 pounds; failed at 1130 pounds. Check-beam, weight, 18 pounds; failed at 1420 pounds. Efficiency of built beam, 80 per cent.

FIG. 17.



CLARK'S BEAM.

No. 9. Exactly the same as No. 8, except that 1.5-inch wire nails were used. Built beam weighed 26 pounds; failed at 1440 pounds. Check-beam weighed 17 pounds; failed at 1480 pounds. Efficiency of built beam, 97.0 per cent.

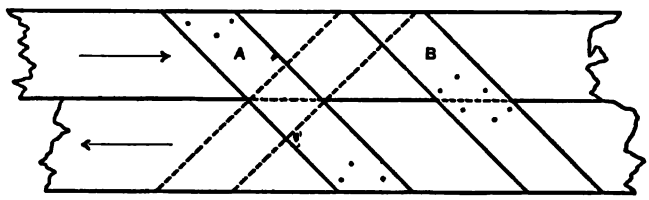
Summary of Tests of Clark's Beam.

Number.	Ratio Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.
1	12	7500	70.7
2	12	5200	67.3
3	12	7800	72.2
4	12	6180	77.6
5	12	6980	72.8
6	12	8400	67.9
		Average.....	71.4
7	24	1680	65.0
8	24	1130	80.0
9	24	1440	97.0
		Average.....	80.7
	Average of	all nine.....	74.5

Conclusions in Regard to Clark's Beam.—The tests show conclusively that Clark erred in assuming that the 95 per cent. efficiency obtained from his models is applicable to practical cases. In view of the results of the tests, the writer would not feel safe in assuming the efficiency as more than 65 per cent. for working purposes. The curves in Diagram 4 show that this beam deflects almost double the amount of a solid beam—a defect which the writer has observed in every case where he has seen this beam used in actual practice.

The horizontal unit shearing-stress in the neutral plane of a beam is 1.5 times the mean shearing-stress in a vertical cross-section, and is the same at all points if the beam carries a center-load. Since this load varies inversely with the span, it is clear that the unit horizontal shear will be halved, if the length of the beam is doubled. Clark's model was 3 inches deep, and should have been 30 inches long. He, however, put

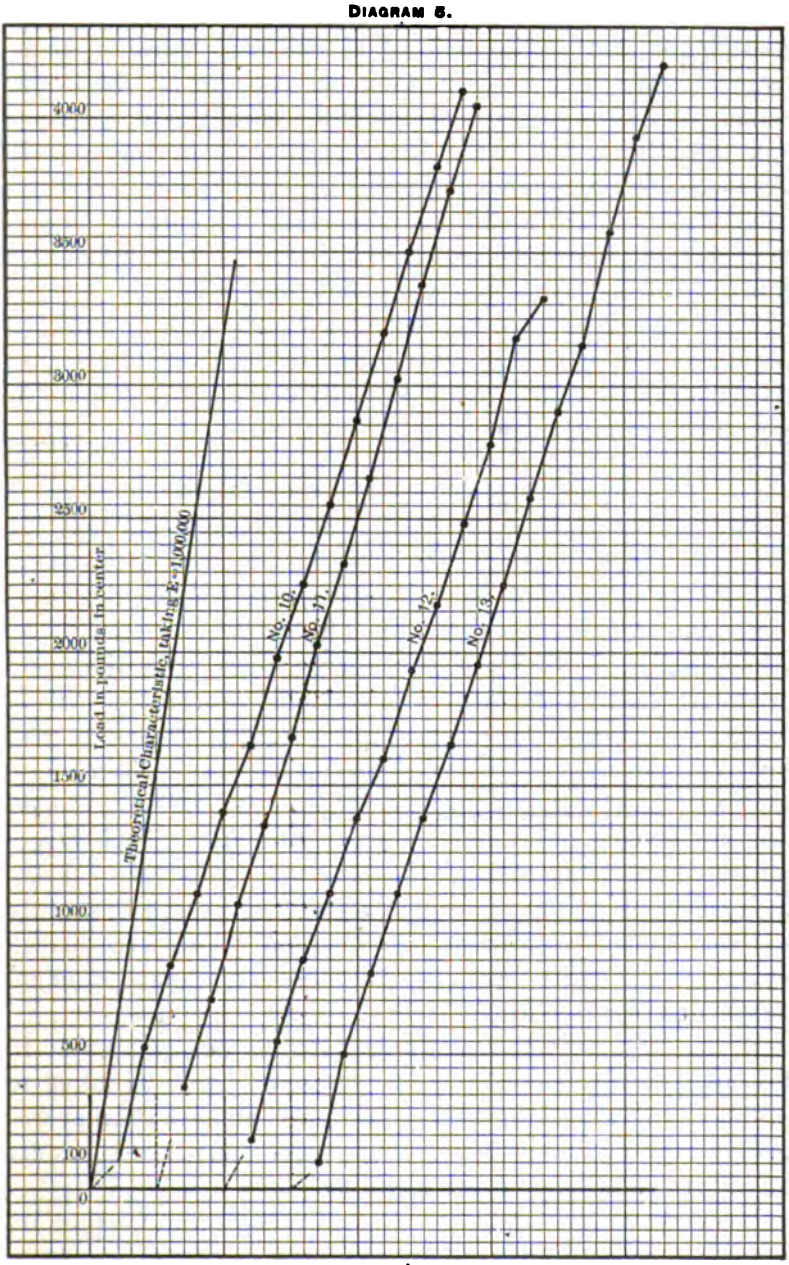
FIG. 18.



it 120 inches long, which makes the ratio of span to depth of beam equal to 40, a proportion no one would dream of using in practice. It is clear that in quadrupling the length he cut down the unit shear in the joint to one-quarter of its former value. Hence his value for the efficiency is worthless for practical cases. The ratio of span to depth in the beams here tested is 12 and 24. The average results for the former are 74.5, and the latter increase to 80.7, for the reason just shown.

Furthermore, Clark errs in saying "The strength of a girder built up in this way will not be affected by shrinkage, for its resistance is maintained by the board-struts, which act only in the direction of their length." As a matter of fact, the struts act in a wholly different manner. Fig. 14 is traced from Clark's figure and shows his method of nailing the struts. Referring to strip A, Fig. 18, it is clear that if the sticks slide in direction of the arrows, the leverage on the nails will be so great that

DIAGRAM 5.



BEAMS WITH 18 PIPES DRIVEN THROUGH COMPONENT STICKS.

Each horizontal space=1/32 inch deflection.

they will at once draw out, or the piece A will be split. The liability of this will be lessened by nailing as at B, while the best plan is of course to nail at both ends and center of struts, as in Fig. 17. The latter method was used in all the beams tested, yet in every case, long before the beam broke, the struts split open or the nails were drawn partly out, or bent over in the wood, thereby permitting the component beams to slide on each other. It was found that no amount of nailing could prevent this, and that the beams tended to fail by shearing along a line of nail-holes, when too many nails were used.

The writer does not consider this a cheap beam to build; it occupies more space than other forms, owing to the side-struts; its deflection is great; hence it would seem that the beam is not to be recommended.

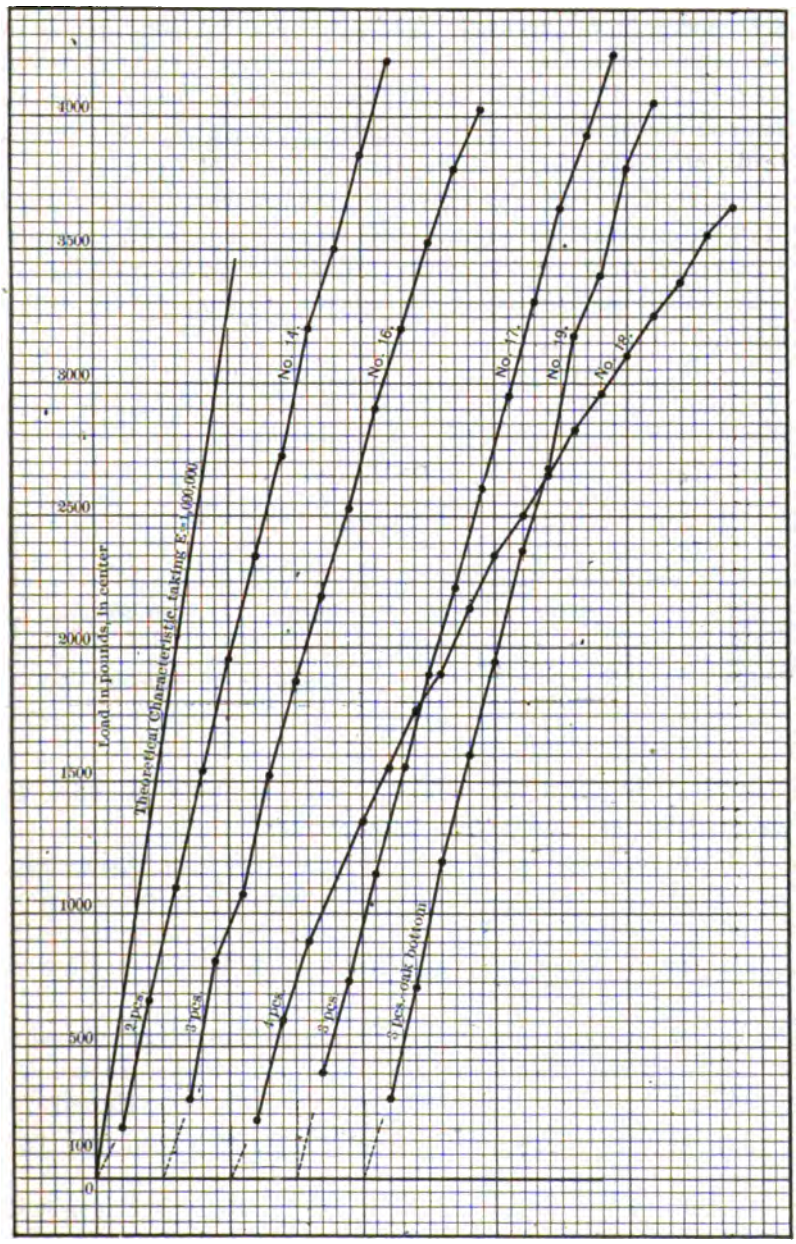
VI. Beams Secured with Pipes and Bolts.

The readiest method of building up a beam is to bolt the component parts together. It is impossible, however, to make an efficient beam with bolts of practicable size, as they bend easily and allow the parts to slide. Bolts large enough to resist bending and offer sufficient bearing-surface for the wood are entirely too costly, and are not readily procurable. It occurred to the writer that the same idea could be cheaply carried out by driving pieces of steam-pipe through holes bored into the beam, and passing through the pipes sufficient bolts to hold the pieces together. Common black three-quarter inch steam-pipe was used in all the experiments; the pieces were cut 0.5-inch shorter than the depth of the beam; one end was slightly rounded on an emery-wheel, and then the pieces were driven by sledge into auger-holes one- and one-sixteenth-inch in diameter, bored into the beam.

Beams Nos. 10 to 13 had 18 pipes driven through them. In this and all other forms yet to be described, care was taken to avoid cutting into the center of the beam, and an attempt was made to cause the fractures to take place outside of the holes or slots used for bolts, pipes, keys, etc.

It was found that when 18 pipes were used, the fracture invariably took place through the pipe-holes. Beams Nos. 14 to 17 inclusive were, therefore, provided with only 12 pipes; and in no case did the line of fracture pass through two consecutive holes as before.

DIAGRAM 6.



BEAMS WITH 12 PIPES DRIVEN THROUGH COMPONENT STICKS.

Each horizontal space = 1/32 inch deflection.

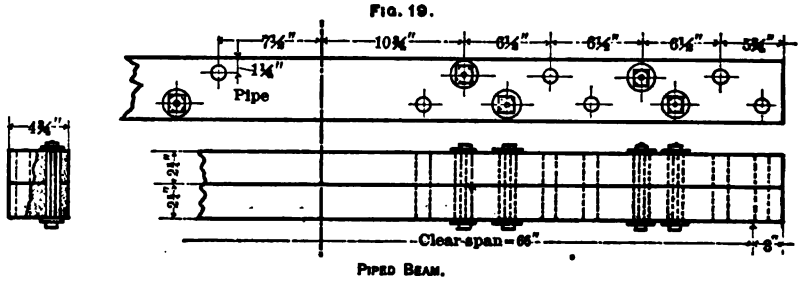
Diagrams 5 and 6 show deflection.

No. 10. Two pieces of white pine, each 2.75 inches deep and 4.75 inches broad, secured by 18 pipes and 8 bolts 0.5-inch in diameter. (See Fig. 19.) Span, 66 inches. Weight of beam, 39 pounds. Broke at 6850 pounds. Check-pieces: A weighed 12 pounds, broke at 2540 pounds; D weighed 13 pounds, broke at 2060 pounds. Efficiency of beam, 74.5 per cent.

No. 11. Everything exactly as in No. 10, except that weight of beam was 41 pounds. Broke at 7600 pounds. Check-pieces: A weighed 14 pounds, broke at 2100 pounds; D weighed 13 pounds, broke at 2540 pounds. Efficiency of beam, 81.9 per cent.

No. 12. Everything exactly as in No. 10, except that weight of beam was 44 pounds. Broke at 3340 pounds. Check-

FIG. 19.



PIPED BEAM.

pieces: A weighed 16 pounds, broke at 1860 pounds; D weighed 14 pounds, broke at 2460 pounds.

Built beam failed in defect in piece of timber on tension-side, located between two pipes; hence this test is of no value except to indicate deflection. The defect was known before the beam was built; and of course no such beam would have been allowed in any practical construction.

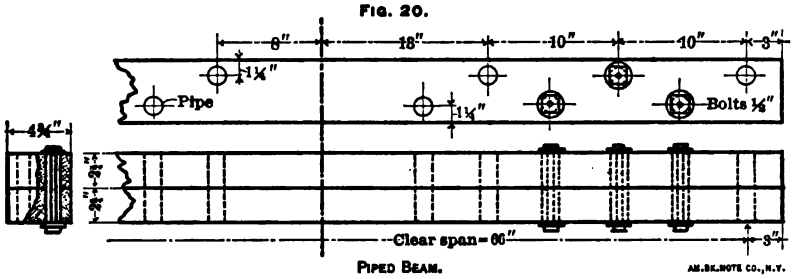
No. 13. Everything as in No. 10, except that weight of beam was 42 pounds. Broke at 8000 pounds. Check-pieces: A weighed 13 pounds, broke at 2020 pounds; D weighed 14 pounds, broke at 2520 pounds. Efficiency of beam, 88.1 per cent.

No. 14. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide, secured by 12 pipes and 6 bolts 0.5-inch in diameter. Span, 66 inches. (See Fig. 20.) Weight of beam, 38 pounds. Broke at 8320 pounds. Check-pieces: B weighed

14 pounds, broke at 2530 pounds; C weighed 14 pounds, broke at 2170 pounds. Efficiency of beam, 88.4 per cent.

No. 15. Two pieces of white oak, each 2.125 inches deep and

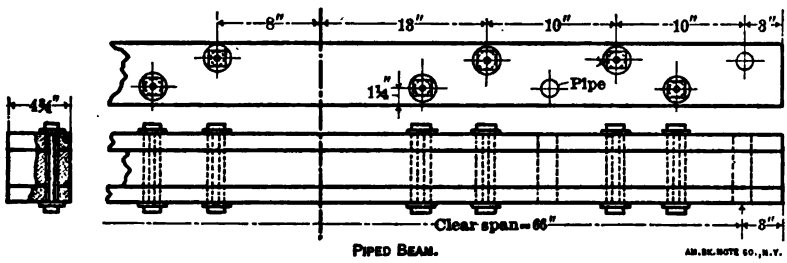
FIG. 20.



PIPED BEAM.

5.5 inches wide, secured by 12 pipes and 6 bolts 0.5-inch in diameter. Span, 66 inches. (See Fig. 20.) Weight of beam, 55 pounds. Broke at 5450 pounds. Check-pieces: A weighed

FIG. 21.

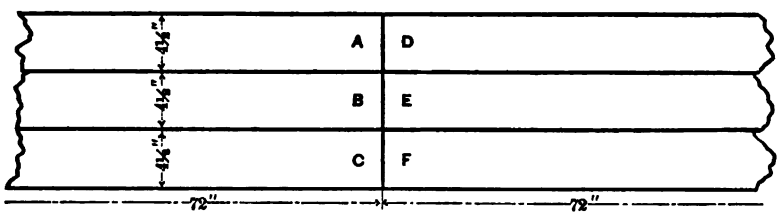


PIPED BEAM.

21 pounds, broke at 1330 pounds; D weighed 21 pounds, broke at 1800 pounds. Efficiency of beam, 86.9 per cent.

No. 16. Three pieces of white pine; center piece 2.75 inches

FIG. 22.



SHOWING METHOD OF CUTTING OUT CHECK PIECES FOR BEAMS NOS. 16 AND 17.

deep and 4.75 inches wide; top and bottom pieces 1.375 inches deep and 4.75 inches wide; all secured by 12 pipes and 8 bolts 0.5-inch in diameter. Span, 66 inches. (See Fig. 21.) Weight

of beam, 44 pounds. Broke at 7340 pounds. Fig. 22 shows method of cutting out top and bottom pieces, together with small check-pieces, for beams Nos. 16 and 17. Check-pieces: B weighed 17 pounds, broke at 2100 pounds; E weighed 8 pounds, broke at 600 pounds.

It should be noted that in this beam and No. 17 the central piece failed first. It is difficult to say what is the proper method of computing the efficiency. As the central piece failed first, it seems that the strength of B, the check-piece for A, should be used as a basis.

This will give:

$$\text{Efficiency of beam} = \frac{100 \times (7340 + 22)}{(4 \times 2100) + (2 \times 17)} = 87.3 \text{ per cent.}$$

We might, however, be justified in using as basis the values derived from both check-pieces, B and E. Doing this,

E broke at 600 pounds, and weighed 8 pounds; hence a piece twice as thick would carry $4 \times 604 = 2416$ pounds.

B weighed 17 pounds, and broke at 2100 pounds. Hence,

$$\text{efficiency of beam} = \frac{100 \times (7340 + 22)}{2 \times (2416 + 2109)} = 81.3 \text{ per cent.}$$

Either method is sufficiently accurate, since the average of values given for Nos. 16 and 17 by each method is about the same.

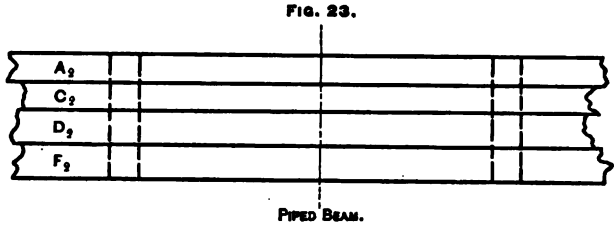
No. 17. Everything exactly as in No. 16, except that beam weighed 42 pounds. Broke at 7960 pounds. Check-pieces: B weighed 8 pounds, failed at 520 pounds; D weighed 14 pounds, failed at 2400 pounds. Efficiency: computing the efficiency with check-piece D as basis gives $\frac{100 \times (7960 + 21)}{4 \times 2400 + 2 \times 14} = 82.9$ per cent.

If figured on B and D jointly, as in the preceding case, the result is an efficiency of 88.6 per cent.

No. 18. Four pieces of white pine, each piece 1.375 inches thick and 4.75 inches wide, superimposed and secured by 12 pipes and 8 bolts 0.5-inch in diameter. Span, 66 inches. (See Figs. 21 and 23.) Weight of beam, 40 pounds. Broke at 6000 pounds. All the pieces were laid out as in Fig. 22, and those in center were used as check-pieces. Check-pieces: B_2 weighed 7 pounds, broke at 600 pounds; E_2 weighed 7 pounds, broke at 750 pounds. Efficiency of beam, 55.5 per cent.

No. 19. Beam composed of one piece of white pine 0.875-inch thick on top, another piece of white pine 2.75 inches thick in center, and a piece of oak 1.875 inches thick on bottom, or tension-side. All pieces 4.75 inches wide. Secured by 12 pipes and 8 bolts, as in Fig. 24. Span, 66 inches. Weight of beam, 50 pounds. Broke at 9140 pounds. Check-piece: D weighed 16 pounds, failed at 2350 pounds. Efficiency (referred to D), 97.2 per cent.

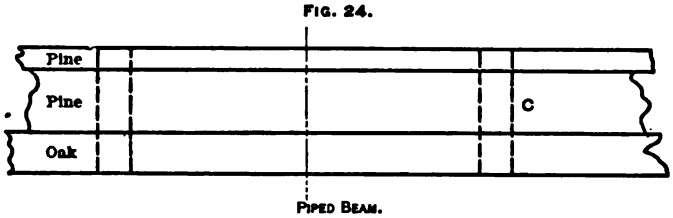
FIG. 23.



The results of the first four show that too many pipes were used. Even with this defect the efficiency exceeds that of Clark's beam of the same size by 10 per cent. But the deflection is about the same—double that of a solid beam. It was noted during the tests that the pine would split opposite the pipes and tend to open, thereby permitting sufficient slippage to cause the large deflection.

The fact that in a solid beam the shearing-stress decreases

FIG. 24.



(though not in direct proportion) with the distance from the neutral plane suggested the form of beam used in Nos. 16 and 17, the idea being to supply solid timber in the neutral axis, and make the joints where the shearing-stresses would be smaller. The resultant efficiencies do not show any advantage gained by this method. It was apparent, in all the tests made on beams consisting of more than two pieces, that the slight yielding of the wood around the fastenings allowed suffi-

Summary and Conclusions Regarding Piped Beams.

Number of Beam.	Number of Pieces Used.	Number of Pipes Used.	Ratio Span to Depth.	Breaking Load. Pounds.	Efficiency. Per Cent.
10	2	18	12	6850	74.5
11	2	18	12	7600	81.9
12	2	18	12	3340	Flaw.
13	2	18	12	8000	83.1
				Average...	81.5
14	2	12	12	8320	88.4
15	2	12	15.5	5450	86.9
				Average...	87.7
16	3	12	12	7310	87.3
17	3	12	12	7960	82.9
				Average...	85.1
16	Efficiencies by alternate method.			7340	81.3
17				7960	88.6
				Average...	85.0
18	4	12	12	6000	53.5
19	3	12	12	9140	97.2

cient slipping of the pieces to change the conditions found in a solid beam. The component parts acted more or less independently, each piece in some of the tests showing a very distinct neutral axis of its own. The writer's conclusion is that with this method of construction three pieces may be as good as two, but no better. The deflection is about the same in each case.

The writer cannot advocate the use of piped beams. A careless workman would almost invariably bore the holes too large, to save exerting himself when driving the pipes, and this defect would be fatal to stiffness and strength.

No. 18 shows clearly the effect of the slipping in the center added to that in the other two joints. This beam has little strength or stiffness and should be avoided.

The old builders strongly advocated putting oak on the tension-side of a built beam. This was done in No. 19. The efficiency, referred to pine alone, is quite high, but not high enough, in the writer's opinion, to compensate for the extra cost of using the oak. The stiffness is very little greater than all-pine. Part II. shows that little is gained by the use of oak.

It would seem that a better plan would be to use slightly larger pine beams.

VII. Beams with Hard-Wood Keys or Joggles.

This form of beam has long been used. Tredgold states that the joint depth of all the keys should amount to one and one-third times the total depth of the beam, and the breadth of each key should be twice its depth. All other writers since the time of Tredgold have contented themselves with copying this rule. It is also recommended by some that the grain of the keys should run vertically. Others state that the depth of the keys must be deducted from the total depth of the two timbers, in order to obtain the effective depth to be used in calculating the strength. How much reliance is to be placed on these statements will appear from the tests.

Keys in the beams under test were made of thoroughly seasoned oak, the grain running lengthwise instead of vertically, as suggested above. Each key was in two parts, cut taper, and these parts were driven firmly home with a heavy mallet.

A defect urged against this beam is the liability of keys to loosen through shrinkage. The writer considers this defect mostly imaginary, if the keys themselves are thoroughly seasoned. The material in the beam will in that case rather tend to grip the keys tighter in case it shrinks.

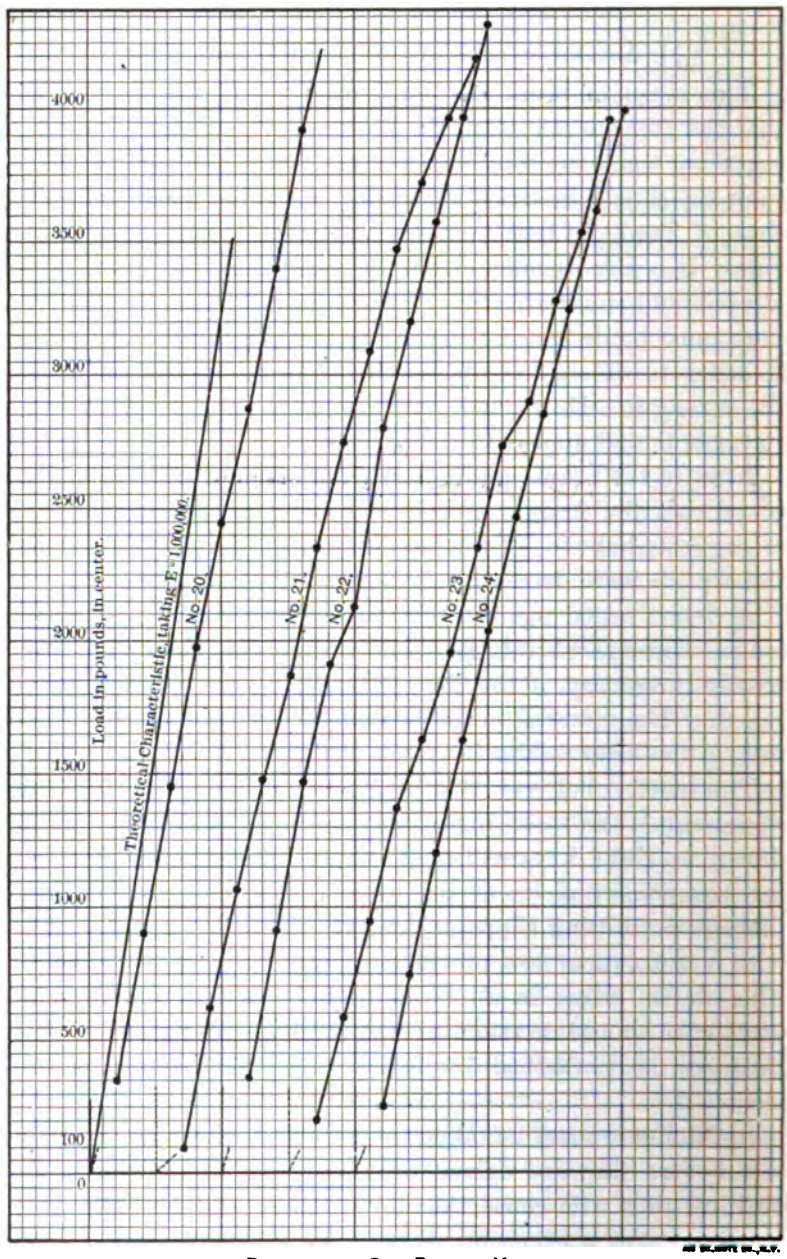
The writer believes that the greatest evil resulting from shrinkage is the loosening of the bolts. Unless the bolts are drawn up tight, keys will tend to turn around in the slots, and the component pieces of the beam will ride up on the keys, thereby permitting slippage along the joint between the sticks, with a consequent loss of efficiency and increase of deflection. This defect exists in all forms of built beams, and therefore extra care should be taken to jam the bolts up tight, and keep them so by periodical inspection, until the timber is fully seasoned.

Special care must be taken to prevent the keys from getting wet before they are driven in place, since it is evident that such wetting will cause the key to swell and act like green wood. Large washers on all bolts are imperative.

Diagram 7 shows deflections.

No. 20. Two pieces of white pine, 2.75 inches deep and 4.75

DIAGRAM 7.



BEAMS WITH OAK DRIVING KEYS.

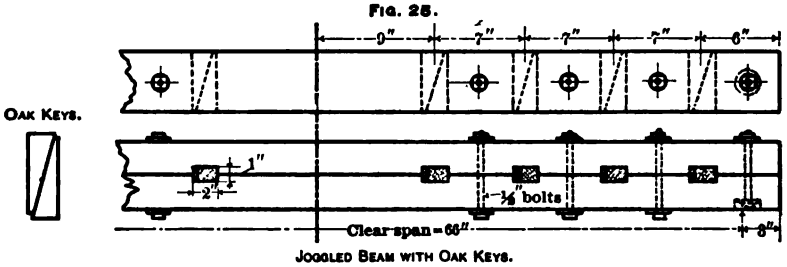
Each horizontal space = 1/32 inch deflection.

inches wide, secured by eight 0.5-inch bolts and eight oak keys 1 by 2 inches. Span, 66 inches. (See Fig. 25.) Weight of beam, 40 pounds. Broke at 10,600 pounds. Check-pieces: B weighed 17 pounds, broke at 2730 pounds; C weighed 17 pounds, broke at 3100 pounds. Efficiency of beam, 90.8 per cent.

No. 21. Everything exactly as in No. 20, except that beam weighed 39 pounds. Broke at 6800 pounds. Check-pieces: A weighed 15 pounds, broke at 1920 pounds; D weighed 16 pounds, broke at 2190 pounds. Efficiency of beam, 82.6 per cent.

No. 22. Everything exactly as in No. 20, except that beam weighed 37 pounds. Broke at 8400 pounds. Check-pieces: A weighed 14 pounds, broke at 2100 pounds; D weighed 16

FIG. 25.



JOGGLED BEAM WITH OAK KEYS.

pounds, broke at 2330 pounds. Efficiency of beam, 94.7 per cent.

No. 23. Everything exactly as in No. 20, except that beam weighed 36 pounds. Broke at 5260 pounds. Check-pieces: A weighed 16 pounds, broke at 2270 pounds; D weighed 14 pounds, broke at 2150 pounds.

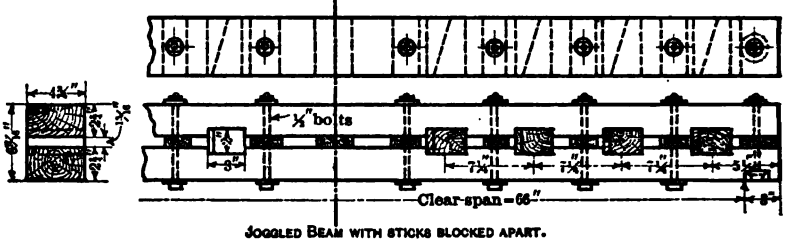
This beam failed in a previously known flaw on the tension-side before there was any appreciable stress brought to bear on the keys. The friction of the pieces caused by the pressure due to bolts took up most of the stress. The failure of the beam is due to the flaw alone, hence the breaking-strength in this case affords no basis for computing efficiency of the joint. The test is reported merely for sake of the deflection-curves.

No. 24. Everything exactly as in No. 20, except that beam weighed 36 pounds. Broke at 8660 pounds. Check-pieces:

B weighed 16 pounds, broke at 1780 pounds; C weighed 18 pounds, broke at 2200 pounds. Efficiency of beam, 108.5 per cent.

This high efficiency is probably due to the fact that check-piece B was the weakest of the four pieces cut out of the same plank; still, there was no apparent difference between A and B. This case shows the impossibility of getting material perfectly homogeneous throughout.

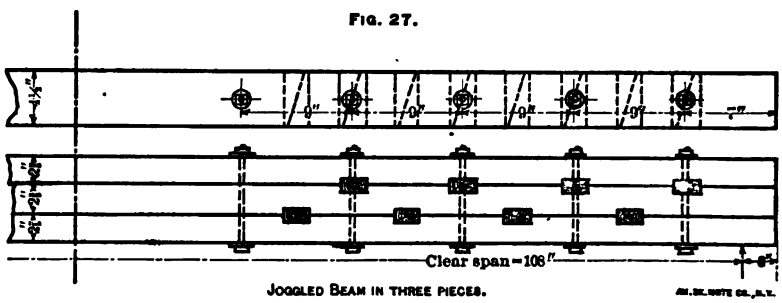
FIG. 26.



JOGGLED BEAM WITH STICKS BLOCKED APART.

No. 25. Everything exactly the same as No. 20, except that keys were 3x2 inches and the two sticks were held fifteen-sixteenths of an inch apart in center by Norway-pine strips. (See Fig. 26.) Weight of beam, 40 pounds. Broke at 8910 pounds.

FIG. 27.

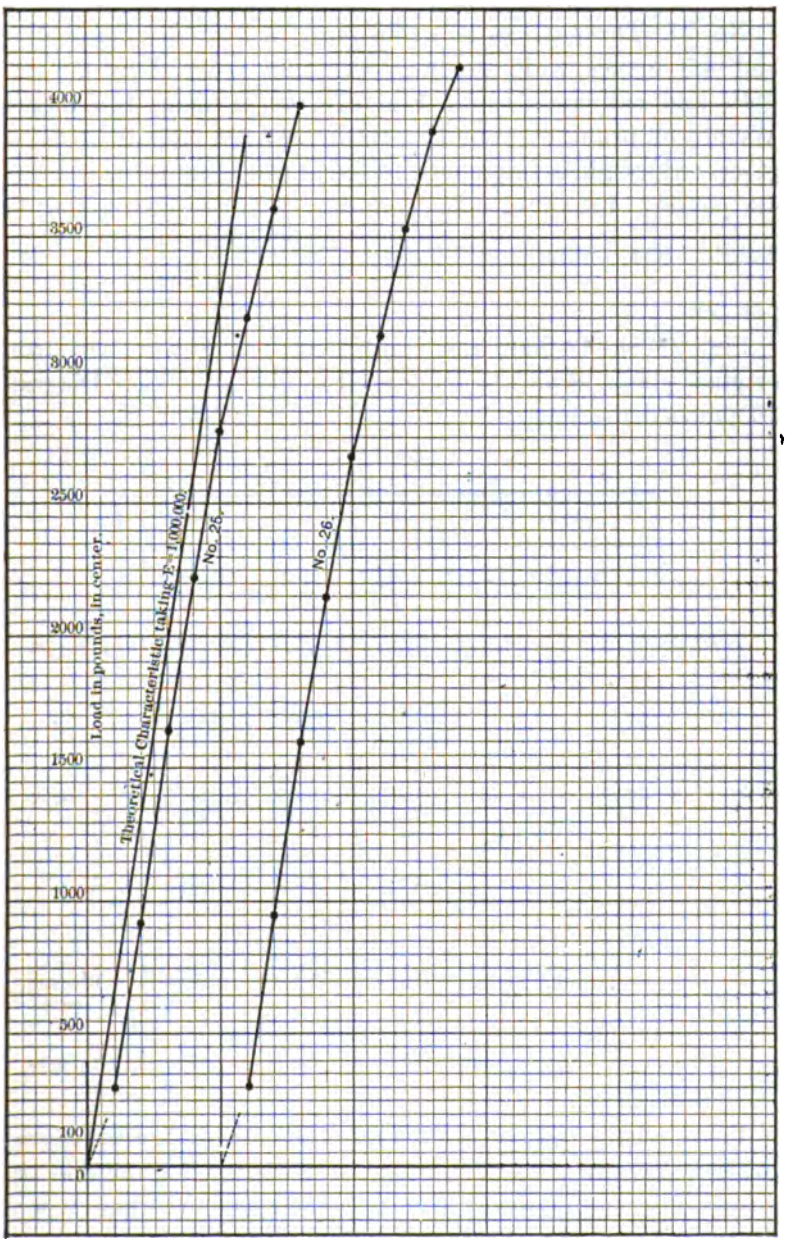


JOGGLED BEAM IN THREE PIECES.

Check-piece weighed 14 pounds, broke at 2400 pounds. Only one was used, the other having had a flaw, due to use of cant-hook. Efficiency of built beam (on basis of solid beam 5.5 inches deep), 92.8 per cent. Diagram 8 shows deflections for beams Nos. 25 and 26.

No. 26. Exactly like No. 25, except that beam weighed 38 pounds. Broke at 7170 pounds. Check-pieces: A weighed 12 pounds, broke at 2220 pounds; B weighed 12 pounds,

DIAGRAM 8.



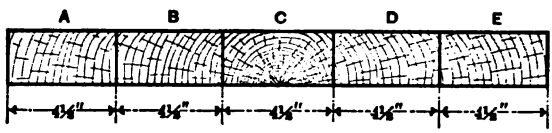
BEAMS WITH OAK DRIVING KEYS, COMPONENT STICKS BEING BLOCKED $\frac{15}{16}$ INCH APART.
Each horizontal space = $\frac{1}{32}$ inch deflection.

broke at 2120 pounds. Efficiency of built beam (on basis of solid beam 5.5 inches deep), 82.6 per cent.

No. 27. Three pieces of white pine, each 2.375 inches deep and 4.5 inches wide, superimposed and secured by 0.5-inch bolts and oak keys, as in Fig. 27. Clear span, 108 inches. Weight of beam, 60 pounds. Broke at 5400 pounds. Check-pieces: B weighed 18 pounds, broke at 870 pounds; D weighed 18 pounds, broke at 900 pounds. Average, corrected for weight, 894 pounds. Efficiency $= \frac{100\ (5400 + 30)}{9 \times 894} = 67.5$ per cent.

Fig. 28 shows how the component pieces were cut from original plank, B and D being used as check-pieces. Diagram 10 shows deflections.

FIG. 28.



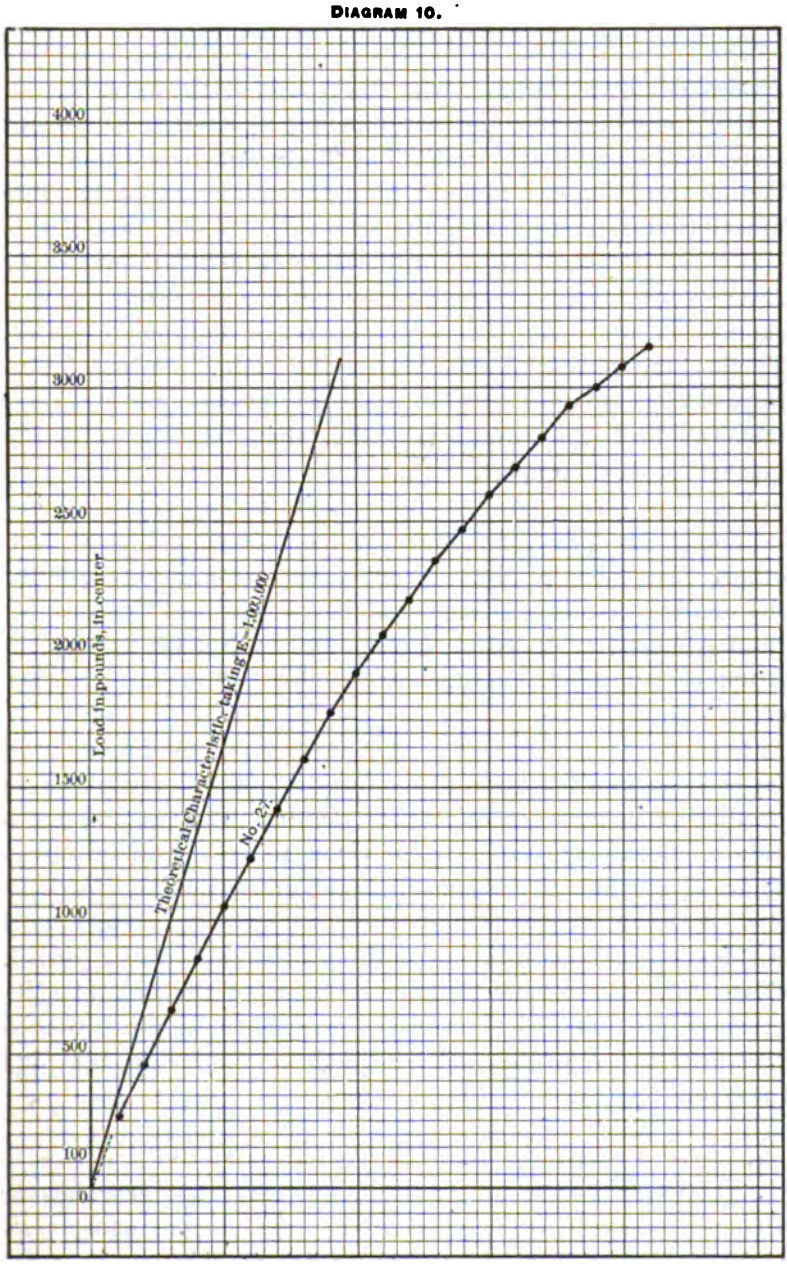
Summary and Conclusions Regarding Beams with Oak Keys.

Number.	Ratio Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.	Remarks.
20	12	10,600	90.8	2 pieces.
21	12	6,800	82.6	2 "
22	12	8,400	94.7	2 "
23	12	5,260	Flaw.	2 "
24	12	8,660	108.5	2 "
		Average...	94.1	
25	12	8,910	92.8	2 pieces. Blocked apart.
26	12	7,170	82.6	
		Average...	87.7	
27	15.2	5,400	67.5	3 pieces.

These results clearly demonstrate that this design is superior to those already seen. The curves in Diagram 7 show that this beam deflects only about 20 to 25 per cent. more than a solid beam.

With regard to the statement that the depth of one key shall be deducted from the total depth of the beam to find the

DIAGRAM 10.



BEAM BUILT OF THREE SUPERIMPOSED STICKS, WITH OAK DRIVING KEYS.
Each horizontal space=1/10 inch deflection.

efficiency, it is to be observed that in this case the resultant depth would be 4.5 inches, and the efficiency $\left(\frac{4.5}{5.5}\right)^2 = 0.669$, a value wholly at variance with the results of the tests.

It is commonly thought that blocking apart the component pieces will increase the strength of the beam, since it increases the apparent depth. Tests on Nos. 25 and 26 show that this procedure does not increase the efficiency. The reason of this is that the increased height of the keys augments the tendency of the keys to roll or overturn in their slots, and this turning cannot be prevented by any amount of bolting of pieces. Further, any beneficial effect due to the friction between superimposed pieces is lost by blocking these pieces apart. Under moderate loads the stiffness is greater than that shown by the beams not blocked apart.

No. 27 confirms the view that in spite of any keys or other fastenings the pieces act more or less independently of each other. The effect of this will, of course, be more apparent as the number of pieces is increased, as is shown in this case by the low efficiency of 67.5 per cent., and by No. 18, which consisted of four pieces, and gave an efficiency of only 55.5 per cent.

The writer does not believe that there is any solid foundation for the idea that the grain of the keys should be vertical. This would greatly increase the cost of making the keys, and probably result in causing many of them to split while driving. There is no difficulty whatever in so proportioning the keys that shearing across is impossible. The main defect with wooden keys comes from an entirely different source. In all of the tests it was noticed that resistance of the oak to side-pressure was much less than the resistance of pine to end-pressure; hence, as the load on the beam increased, the pine indented the oak.

VIII. Beams with Cast-Iron Keys.

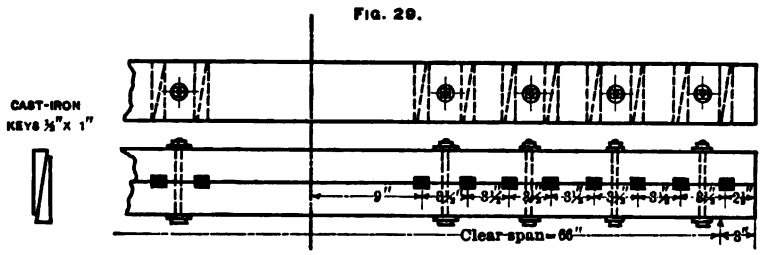
Since there is no wood available for keys, the resistance of which to side-pressure is equal to the resistance of white pine to end-pressure, it is clear that, without using an excessive number of keys, indentation cannot be prevented, and the component parts of the beam must slip. This trouble will be ag-

gravated in proportion as a harder wood is used for the beams. The only remedy is to use iron keys. Such keys were used a century ago in building up the overhead beam of steam-engines, but were abandoned in favor of wood, because they seemed to work loose under the reversals of stress in the beam.* Though their use in railway-stringers is quite common, the writer has never seen them used in beams for buildings. It is evident that the objection noted above would not exist in this case, as the stresses would not be reversed.

Accordingly, two beams were built with ordinary wedge-shaped cast-iron keys and tested. Diagram 9 shows deflections.

No. 28. Two pieces of white pine, 2.75 inches deep and 4.75 inches wide, secured by eight bolts, 0.5-inch in diameter, and

FIG. 29.



JOGGLED BEAM WITH CAST-IRON DRIVING KEYS.

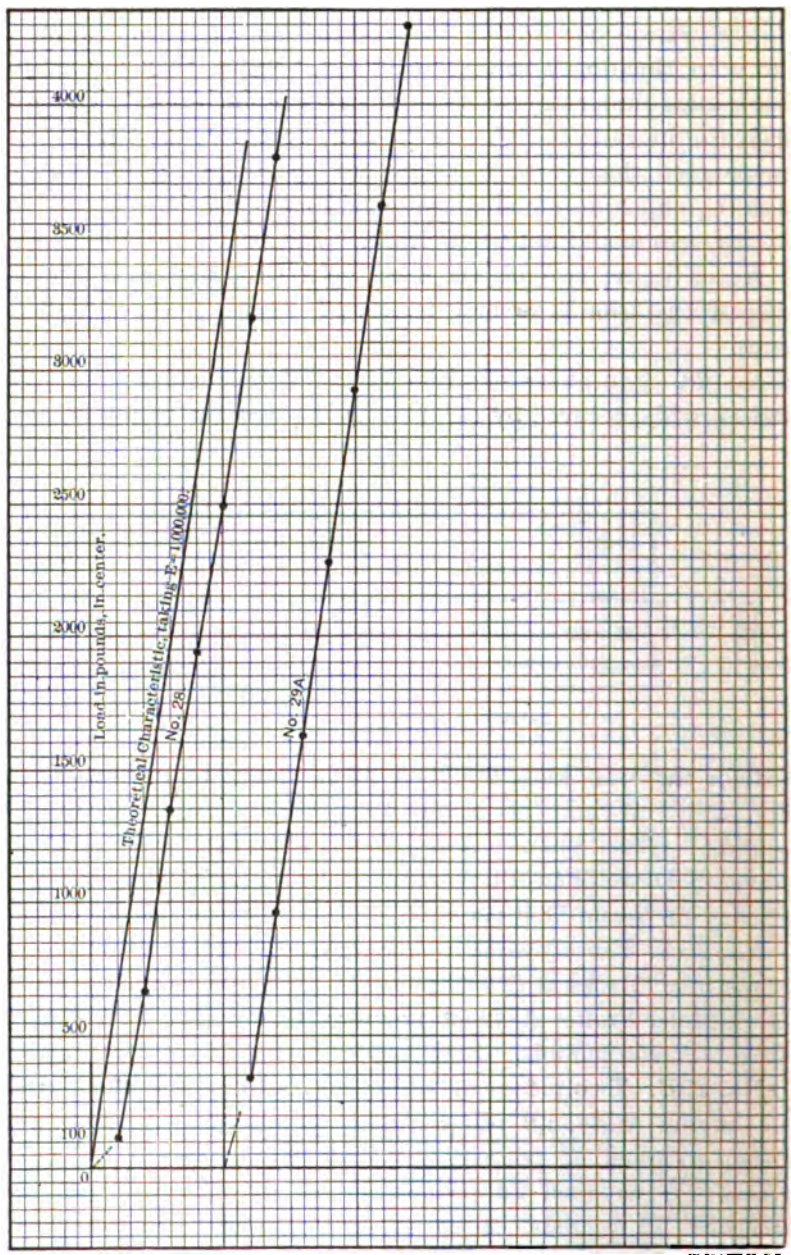
sixteen cast-iron keys, 0.5 by 1 inch. Span, 66 inches. (See Fig. 29.) Weight of beam, 48 pounds. Broke at 10,190 pounds. Check-pieces: A weighed 14 pounds, broke at 2930 pounds; D weighed 14 pounds, broke at 2800 pounds. Efficiency of beam, 88.9 per cent.

No. 29A. Exactly same as No. 28, except that beam weighed 50 pounds. Broke at 11,190 pounds. Check-pieces: B weighed 14 pounds, broke at 3050 pounds; C weighed 16 pounds, broke at 2700 pounds. Efficiency of beam, 97.3 per cent.

No. 29B. Two pieces of white oak, 2.75 inches thick and 4.5 inches wide, all other details precisely as in Fig. 29. Beam weighed 62 pounds, broke at 16,050 pounds. Check-pieces: A weighed 22 pounds, broke at 4300 pounds; D weighed 22 pounds, broke at 4650 pounds. Efficiency of beam, 89.6 per cent. Deflection not taken.

* *Encyclopædia Britannica.* Article, "Building."

DIAGRAM 9.



BEAMS WITH CAST-IRON DRIVING KEYS.
Each horizontal space = 1/32 inch deflection.

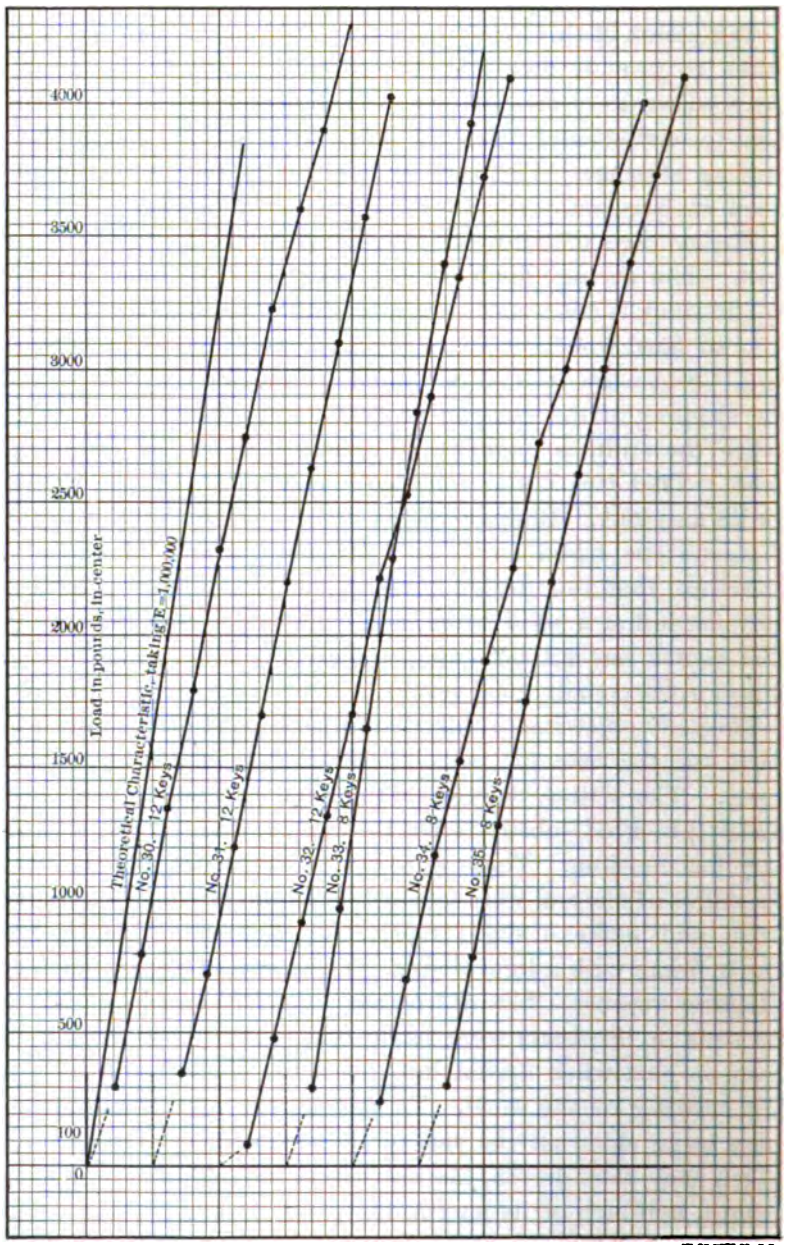
Conclusions Regarding Beam with Plain Cast-Iron Keys.

The action of this beam under test showed that care must be taken to provide ample-sized washers on the bolts. In all cases where keys are used the component parts of the beam tend to separate and rotate the keys in their slots, thereby permitting slippage. The remedy for this is the use of ample-sized washers, rather than a large number of bolts.

The average efficiency of Nos. 28, 29A and 29B is 91.9 per cent. The average for Nos. 20 to 24 is 94.1 per cent. So far as efficiency is concerned, it matters little whether the keys be iron or hard wood, *provided a sufficient number of wooden keys are used.* Since the iron keys cannot be indented by the wood, the slipping of the component parts of the beam is reduced; hence the stiffness ought to be greater than in the preceding cases. That this is so is clearly demonstrated by the curves for 28 and 29A. Both curves are almost exactly parallel to the characteristic based on E as 1,000,000 pounds. It appears, therefore, that this design gives a compound beam which is fully as stiff as a solid beam, and over 90 per cent. as strong. The writer has not been able to find any other construction which gives equally good results.

The only objection to this design is the cost of the cast-iron keys, and the time one usually will have to wait to have them made. The writer, therefore, attempted to find a cheap and readily-procured substitute for them, by trying pieces of ordinary flat bar-iron. These were cut to lengths of 4.75 inches; the edges of one end were touched on an emery-wheel to remove burrs and make a slight taper, and they were then driven firmly home with a light sledge. These pieces can be made at a trifling expense. Size of iron bar was 0.375 by 1 inch; hence the slots in the beam were made 0.25-inch wide and 0.5-inch deep, so that when driven into place the keys would have a firm bearing on the wood. To prevent the keys from tearing out a large sliver on the side of the beam opposite that from which they were driven, the slot was at that side flared out for about a half-inch of its length, the width on the outside being half an inch. This beam is cheaper to construct than any other form the writer has tested, and less care is required in cutting out the slots, since in case the latter do not quite range, the

DIAGRAM 11.



BEAMS WITH FLAT WROUGT-IRON KEYS.
Each horizontal space = 1/32 inch deflection.

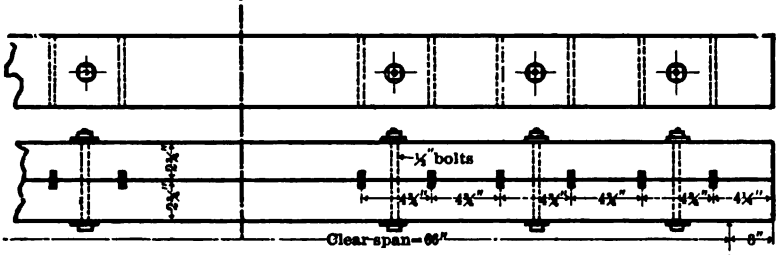
keys, acting like nails, owing to their tapered ends, correct the error when driven into place. Diagram 11 shows deflections.

In some other forms, wrought-iron 0.75-inch square was used for keys, as this shape is less liable to rotate than the narrow and deep keys. Diagram 12 shows deflections of this form of beam.

IX. Beams with Wrought-Iron Keys.

No. 30. Two pieces of white pine, each 2.75 inches deep and 4.75 inches wide, united by six 0.5-inch bolts and 12 flat

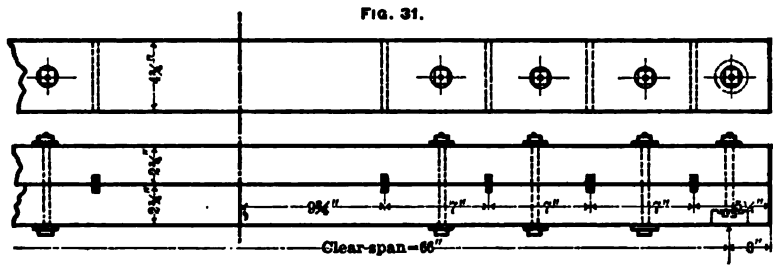
Fig. 30.



Joggled Beam with Straight Wrought-Iron Keys.

wrought-iron keys. (See Fig. 30.) Span, 66 inches. Weight of beam, 35 pounds. Broke at 7800 pounds. Check-pieces: B weighed 12 pounds, failed at 2450 pounds; C weighed 12 pounds, failed at 2600 pounds. Efficiency of beam, 77.2 per cent.

Fig. 31.

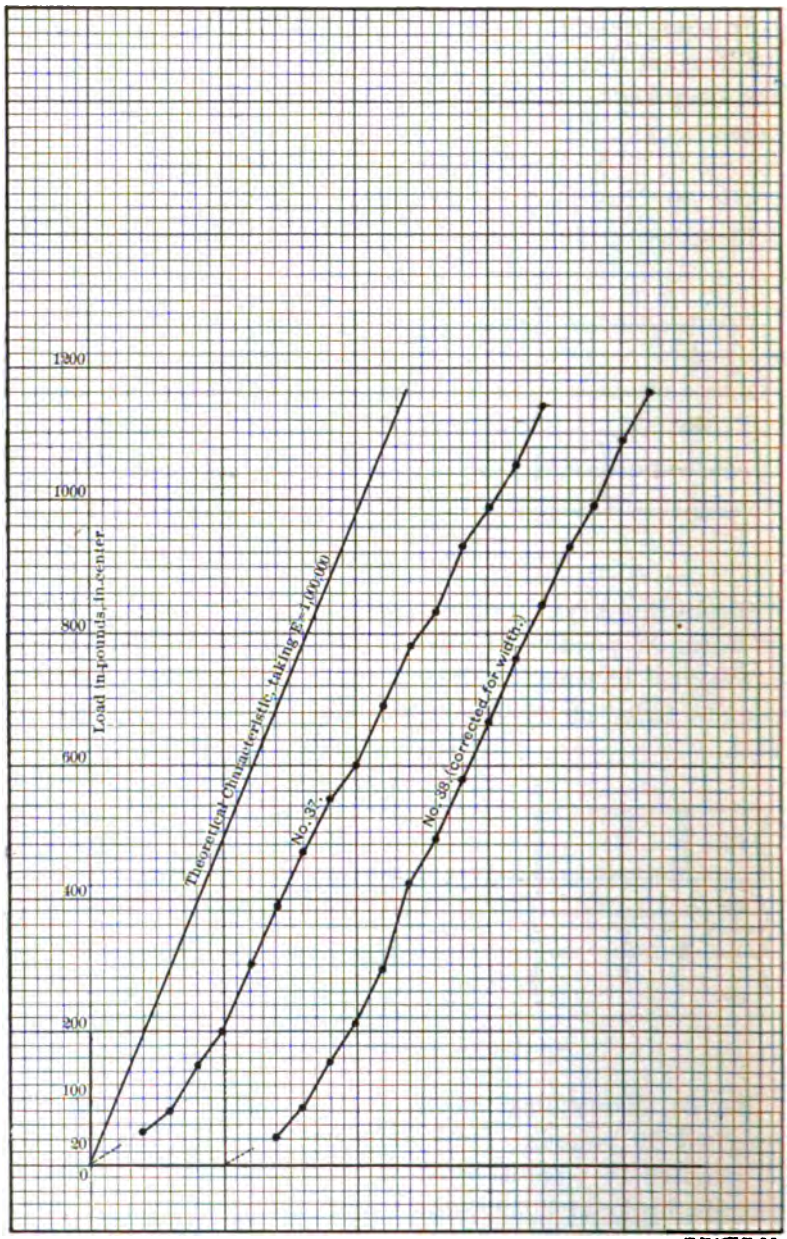


Joggled Beam with Straight Wrought-Iron Keys.

No. 31. Everything exactly as in No. 30, except that beam weighed 40 pounds. Broke at 8200 pounds. Check-pieces: A weighed 16 pounds, broke at 2620 pounds; D weighed 15 pounds, broke at 3030 pounds. Efficiency of beam, 72.3 per cent.

No. 32. Everything exactly as in No. 30. Beam weighed 40 pounds. Broke at 6350 pounds. Check-pieces: B weighed

DIAGRAM 12.

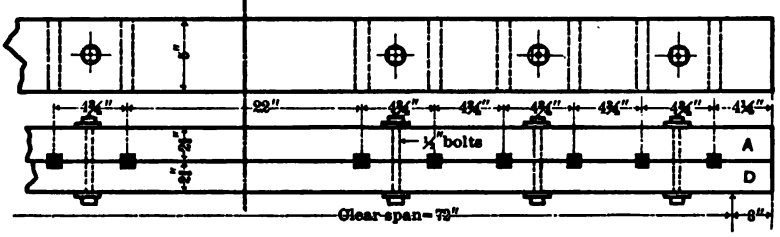


BEAMS WITH ¾" SQUARE WROUGHT-IRON KEYS.
Each horizontal space = 1/32 inch deflection.

15 pounds, broke at 1800 pounds; C weighed 15 pounds, broke at 2200 pounds. Efficiency of beam, 79.3 per cent.

No. 33. Similar to No. 30, except that only 8 keys were used. (See Fig. 31.) Weight of beam, 37 pounds. Broke at 10,750 pounds. Check-pieces: B weighed 15 pounds, broke at 3100 pounds; C weighed 17 pounds, broke at 3480 pounds. Efficiency of beam, 81.6 per cent.

FIG. 32.

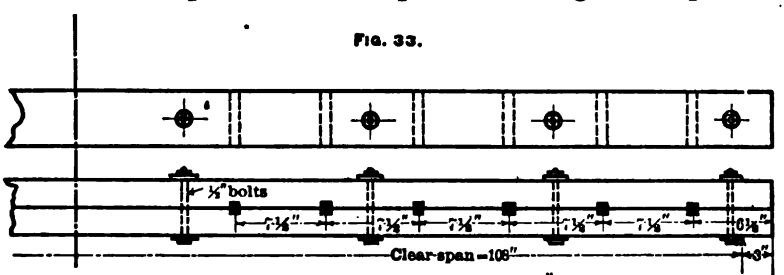


JOGGLED BEAM WITH STRAIGHT WROUGHT-IRON KEYS ¾" SQUARE.

No. 34. Similar to No. 33. Weight of beam, 35 pounds. Broke at 7990 pounds. Check-pieces: A weighed 13 pounds, broke at 2150 pounds; D weighed 14 pounds, broke at 2970 pounds. Efficiency, 78.0 per cent.

No. 35. Similar to No. 33. Weight of beam, 37 pounds. Broke at 9270 pounds. Check-pieces: A weighed 14 pounds,

FIG. 33.



JOGGLED BEAM WITH STRAIGHT WROUGHT-IRON KEYS ¾" SQUARE.

broke at 2530 pounds; D weighed 14 pounds, broke at 2920 pounds. Efficiency of beam, 85.0 per cent.

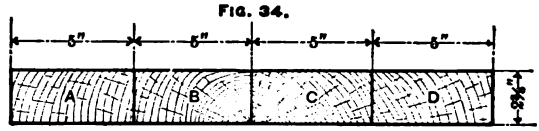
No. 36. Two pieces of white pine, each 2.375 inches deep and 5 inches wide, secured by 6 half-inch bolts and 12 wrought-iron keys cut from bar 0.75-inch square. Span, 72 inches. (See Fig. 32.) Weight of beam, 38 pounds. Broke at 5600 pounds. Check-pieces: B weighed 13 pounds, broke at 1600 pounds; C weighed 13 pounds, broke at 1600 pounds. Efficiency of beam, 87.5 per cent.

No. 37. Two pieces of white pine, 2.375 inches deep and 4.5 inches wide, secured by 8 half-inch bolts and 12 wrought-iron keys cut from bar 0.75-inch square. Span, 108 inches. (See Fig. 33.) Weight of beam, 48 pounds. Broke at 2380 pounds. Check-pieces: A weighed 18 pounds, broke at 620 pounds; C weighed 18 pounds, broke at 730 pounds. Efficiency of beam, 87.8 per cent.

Fig. 34 shows how component sticks and test-pieces were cut from the original plank.

No. 38. Everything as in No. 37, except that beam was 4.25 inches wide and weighed 46 pounds. Broke at 2760 pounds.

FIG. 34.



Check-pieces: B weighed 18 pounds, broke at 720 pounds; D weighed 16 pounds, broke at 790 pounds. Efficiency of beam, 91.1 per cent.

Summary and Conclusions Regarding Beams with Wrought-Iron Keys.

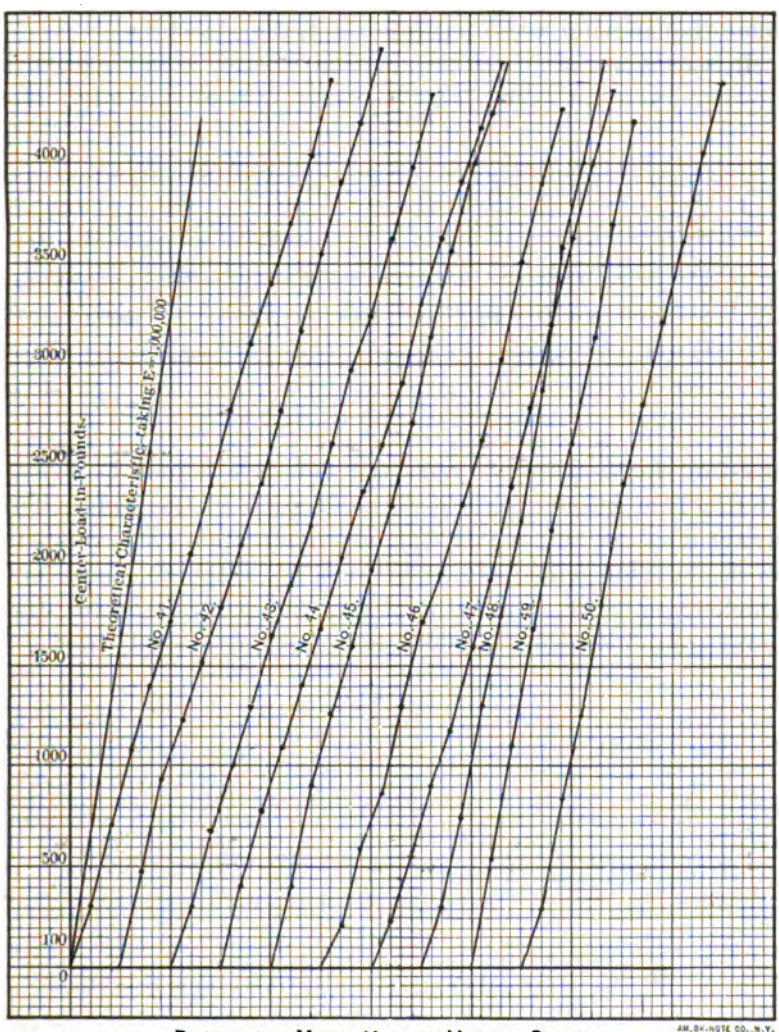
Number.	Ratio Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.	Remarks.
30	12	7,800	77.2	12 keys.
31	12	8,200	72.3	12 "
32	12	6,350	79.3	12 "
		Average...	76.3	
33	12	10,750	81.6	8 keys.
34	12	7,990	78.0	8 "
35	12	9,270	85.0	8 "
		Average...	81.5	
36	15.2	5,600	87.5	12 keys.
37	22.7	2,380	87.8	12 "
38	22.7	2,760	91.1	12 "
		Average...	88.8	

The reason why the first three beams were weaker than the second three is that in the former the keys were 1.75 inches nearer the center of the beam than in the latter.

Owing to the narrow width of the keys the tendency to turn

over and allow the component parts of the beam to slide was very evident. In spite of this, the curves show that the average deflection of these beams did not exceed that of a solid

DIAGRAM 13.



BEAMS WITH METAL KEYS OF VARIOUS SHAPES.

Each horizontal space = ½ inch deflection.

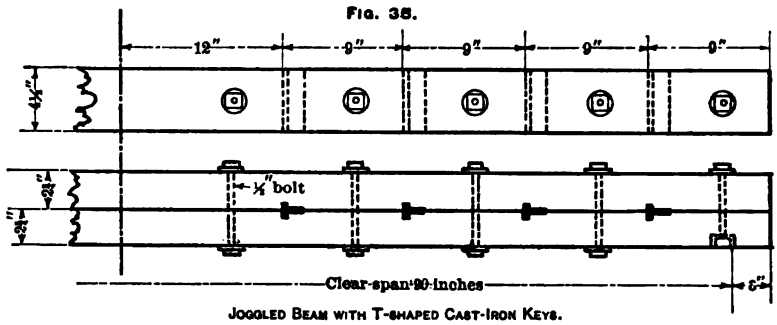
piece by 50 per cent., a better result than that obtained by Clark's construction, or by using pipes.

It was thought that by using square keys the deflection

might be lessened. No. 36 shows, however, about the same rate of deflection as those built with the flat keys. Nos. 37 and 38 show a deflection of about 25 per cent. in excess of a solid piece, if we disregard the initial part of the curves (Diagram 12), which is affected by the fact that the beams were somewhat in wind when put into the testing-machine.

It is evident that these beams deflect more than those with

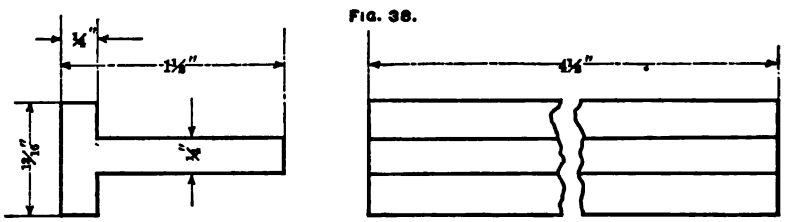
FIG. 35.



JOGGLED BEAM WITH T-SHAPED CAST-IRON KEYS.

east-iron keys, simply because the flat keys cannot be driven up to a firm bearing against the end-wood in the slots. The writer does not advise this style of construction for permanent work, but for temporary structures it is the cheapest and quickest to build, and will answer the purpose better than many complicated forms which have been extensively used.

FIG. 36.



CAST-IRON KEY.

X. BEAMS WITH METAL KEYS OF VARIOUS SHAPES.

After the preceding tests were completed, a summary of the results was published. This led to a good deal of correspondence with parties who had used forms of key not included in the tests. The writer, therefore, pursued the investigation further, with results that will now be given.

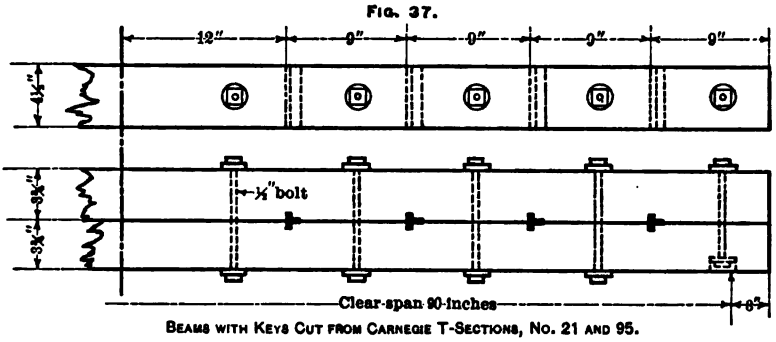
Diagram 13 shows deflection of beams Nos. 41 to 50, inclu-

sive. The writer believes that Nos. 39 to 47 could be made much stiffer by the simple expedient of providing the keys with a taper of about one-eighth of an inch, then driving them firmly into place after the component sticks had been tightly bolted up. In such cases the larger end of the keys should be provided with an ear or lug, arranged so that an ordinary wood-screw can pass through a hole or slot in the lug, screw into the beam, and thereby prevent the key from shaking out of its slot under heavy vibrations, etc.

Beams with T-shaped Keys of Cast-Iron.

No. 39. Two pieces of white pine, 2.75 inches thick and 4.5 inches wide, secured as in Fig. 35. Keys of cast-iron as in Fig. 36. Span, 90 inches. Weight of beam, 44 pounds. Broke

FIG. 37.



BEAMS WITH KEYS CUT FROM CARNEGIE T-SECTIONS, NO. 21 AND 95.

at 5700 pounds. Check-pieces: B weighed 17 pounds, broke at 2360 pounds: C weighed 17 pounds, broke at 2300 pounds. Efficiency of beam, 61.1 per cent.

No. 40. Same as No. 39. Weight of beam, 40 pounds. Broke at 7400 pounds. Check-pieces same as preceding two. Efficiency of beam, 79.3. Average for the two beams, 70.2 per cent. Deflection of these two beams was not noted.

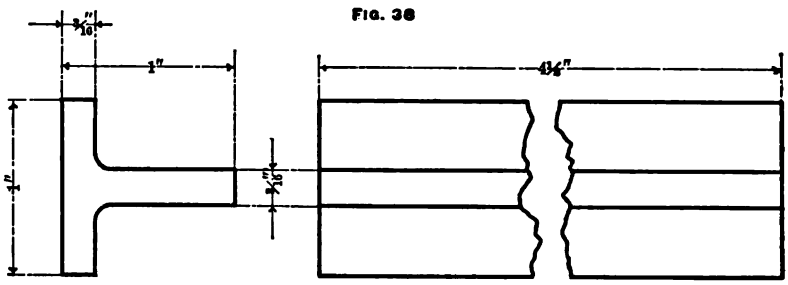
Beams with T-shaped Keys of Steel.

No. 41. Two pieces of white pine, 3.75 inches thick and 4.5 inches wide, secured by bolts and keys made by sawing off pieces of Carnegie T-section, No. 21.* These keys were ground free of burrs on one end, the slots carefully cut to size for a

* See *Pocket Companion.* Carnegie, Phipps & Co. Edition of 1892.

driving fit, and keys were then driven home after the beam was bolted up. (See Fig. 37.) Dimensions of key are shown in Fig. 38. Span of beam, 90 inches. Weight of beam, 58 pounds. Broke at 7900 pounds. Check-pieces: A weighed 23 pounds, broke at 3680 pounds; D weighed 22 pounds, broke at 3700 pounds. Efficiency of beam, 53.6 per cent.

FIG. 38.

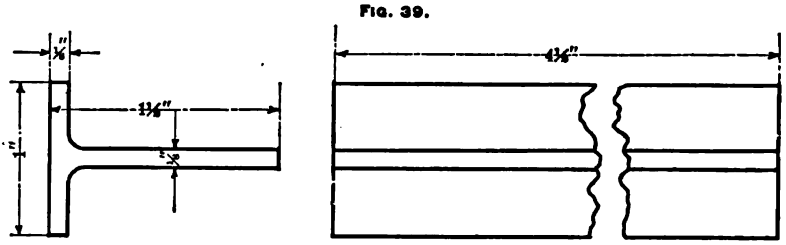


KEY CUT FROM CARNEGIE STEEL SECTION T-21

No. 42. Exactly like No. 41. Beam weighed 60 pounds, broke at 11,100 pounds. Check-pieces: B weighed 25 pounds, broke at 4080 pounds; C weighed 22 pounds, broke at 3800 pounds. Efficiency of beam, 70.4 per cent.

No. 43. Exactly like No. 41, except keys were cut from

FIG. 39.



KEY CUT FROM CARNEGIE STEEL SECTION T-95.

Carnegie T-section, No. 95.* (See Fig. 39.) Beam weighed 56 pounds, broke at 7800 pounds. Check-pieces: B weighed 23 pounds, broke at 3680 pounds; C weighed 22 pounds, broke at 3700 pounds. Efficiency of beam, 51.1 per cent.

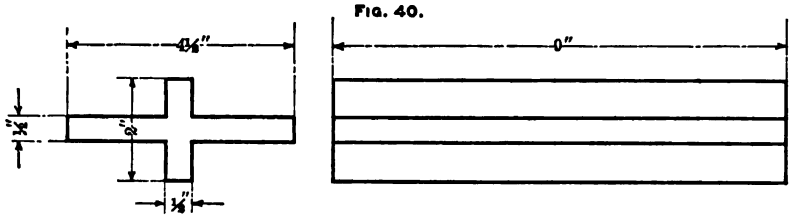
No. 44. Exactly like No. 43. Beam weighed 56 pounds, broke at 10,200 pounds. Check-pieces: A weighed 22 pounds,

* See *Pocket Companion.* Carnegie, Phipps & Co. Edition of 1892.

broke at 3230 pounds; D weighed 20 pounds, broke at 3670 pounds. Efficiency of beam, 73.2 per cent.

The mean of the efficiencies of the four is 62.1 per cent. Nothing further need be said, except that this low result is evidently due to the fact that mere fitting of keys alone will not

FIG. 40.



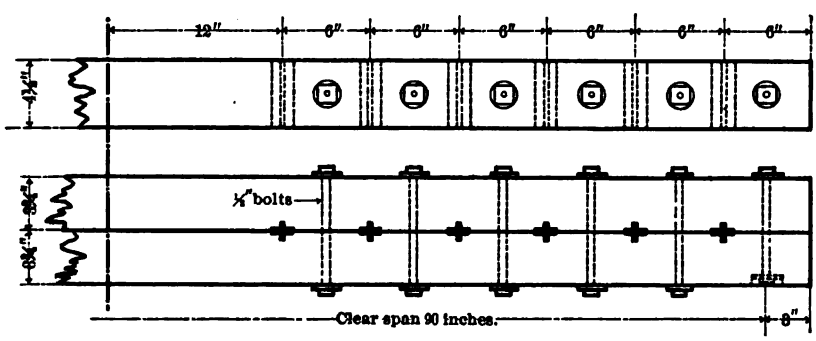
DIMENSIONS OF FULL-SIZED KEY USED ON R. F. AND P. RY. FOR STRINGERS CONSISTING OF TWO PIECES 9"x 15" SUPERIMPOSED.

answer. There must be some means provided for wedging them up.

Beam with Keys of Cross-shaped Section.

The form of cast-iron key used for packing the wooden chords of Howe truss bridges has been used on the Richmond, Fredericksburg and Potomac Railway for keying-up built beams

FIG. 41.

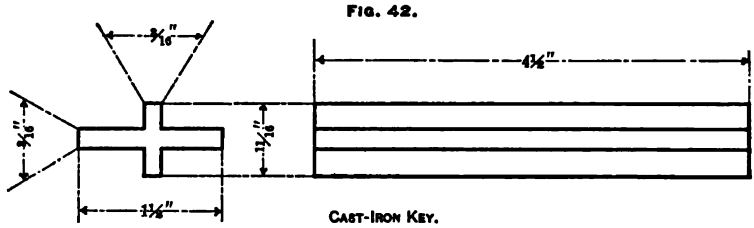


CAST-IRON KEYS OF CROSS-SHAPED SECTION, AS USED ON THE RICHMOND, FREDERICKSBURG AND POTOMAC RAILWAY.

used as bridge-stringers. Fig. 40 shows dimensions of the full-sized keys used for a stringer composed of two superimposed sticks, each 9 inches wide and 15 inches deep; 9 of these being used in a stringer 24 feet long. The writer is indebted to President E. T. D. Myers for drawings and information relative to the practice used on this road.

No. 45. Fig. 41 shows beam, and Fig. 42 the key used. Beam weighed 60 pounds, broke at 11,940 pounds. Check-pieces: B weighed 22 pounds, broke at 3370 pounds; C weighed 20 pounds, broke at 3400 pounds. Efficiency of beam, 88.1

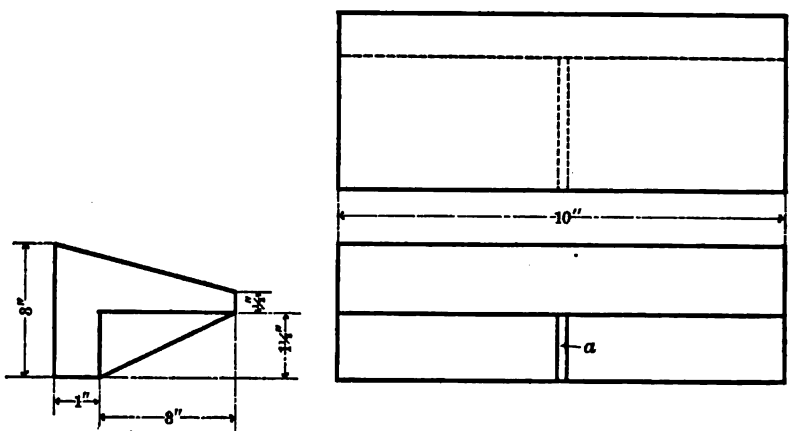
FIG. 42.



CAST-IRON KEY.

per cent. This result is better than that shown by any other form of key not provided with wedges, and is doubtless due to the ample breadth of key which resists the tendency to roll over. The curve shows, however, that the stiffness is inferior to those beams in which wedges were employed.

FIG. 43.



DIMENSIONS OF FULL-SIZED CAST-IRON KEYS USED ON BOSTON AND MAINE RY., FOR STRINGERS CONSISTING OF TWO PIECES 10"X 12" SUPERIMPOSED. THE FIN "a" PREVENTS KEY FROM SHAKING OUT OF ITS SLOT.

The Boston and Maine Key.

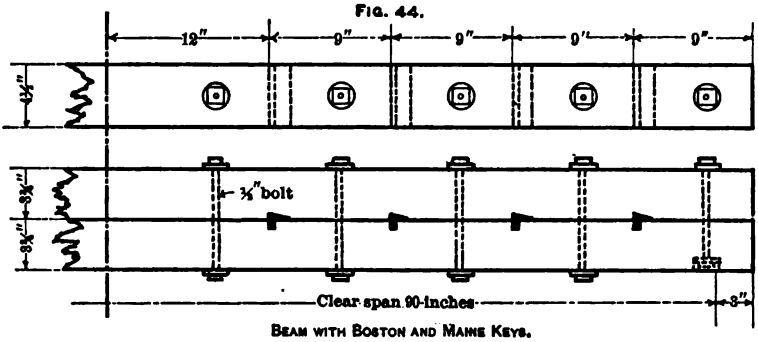
The form of key used in the next two beams is extensively used on the Boston and Maine Railway for building compound stringers. The writer is indebted to J. P. Snow, C.E., of that road, for drawings and other information.* Fig. 43 shows

* *Cf.* "Wooden Bridge Construction on the Boston and Maine Railway," J. Parker Snow, C. E. *Journal of the Association of Engineering Societies* for July, 1895. Detailed information as to method of designing these stringers is given.

actual dimensions of keys, of which 8 are used on a 23-foot span of compound beam composed of two 10- by 12-inch timbers.

No. 46. All dimensions as in Fig. 44, and key like Fig. 45. Beam weighed 54 pounds, broke at 10,630 pounds. Check-beams: A weighed 21 pounds, broke at 4000 pounds; D

FIG. 44.

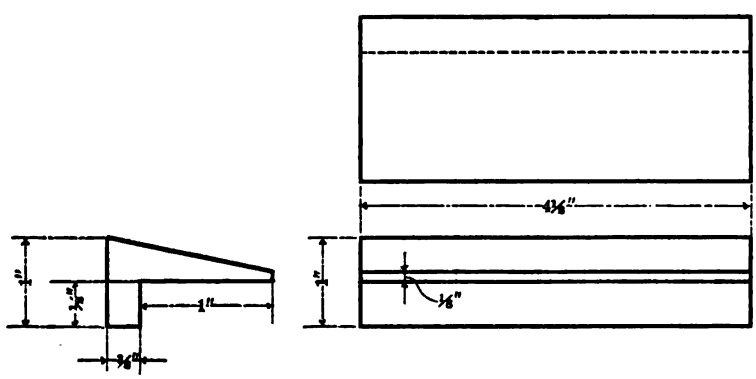


BEAM WITH BOSTON AND MAINE KEYS.

weighed 21 pounds, broke at 3910 pounds. Efficiency of beam, 67.2 per cent.

No 47. Exactly the same as No. 46. Weighed 54 pounds, and broke at 11,130 pounds. Check-pieces, same two as for No. 46. Efficiency of beam, 70.3 per cent. Mean efficiency

FIG. 45.



from the two tests, 68.8 per cent. This efficiency could evidently be increased by making the horizontal stem of the key wider.

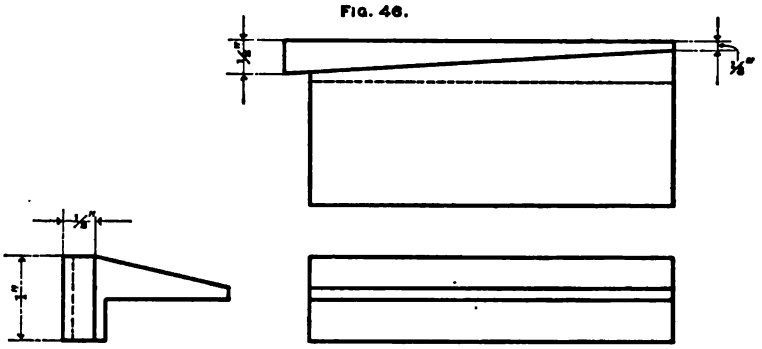
The Boston and Maine Key with Wedges.

It was thought advisable to determine what gain in strength and stiffness could be got by providing some of the preceding

keys with driving-wedges. It will be noticed that by this expedient, both the strength and stiffness were augmented sufficiently to compensate for the additional trouble of putting in the wedges.

No 48. Exactly the same as No. 46, except that keys were of

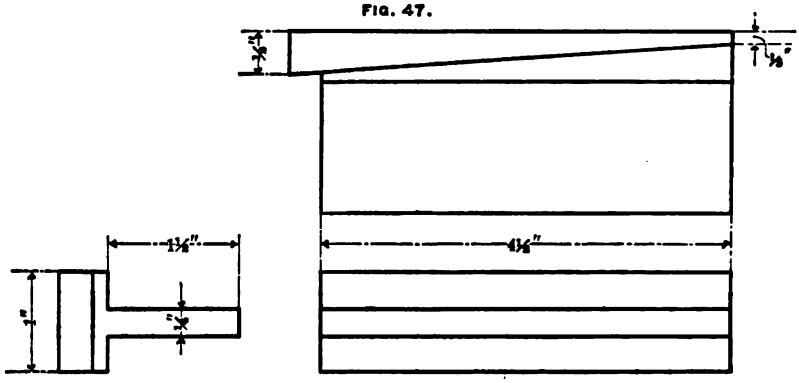
FIG. 46.



MODEL OF BOSTON AND MAINE KEY FITTED WITH WEDGES.

pattern in Fig. 46. Beam weighed 63 pounds, broke at 10,900 pounds. Check-pieces: A weighed 21 pounds, broke at 3450 pounds; D weighed 21 pounds, broke at 3385 pounds. Efficiency of beam, 79.7 per cent.

FIG. 47.



CAST-IRON KEY WITH WEDGE.

T-shaped Keys with Wedges.

No. 49. Exactly the same as No. 41, except that keys were like Fig. 47. Beam weighed 58 pounds, broke at 9600 pounds. Check pieces: A weighed 21 pounds, broke at 3600 pounds; D weighed 20 pounds, broke at 3550 pounds. Efficiency of beam, 67.2 per cent.

No. 50. Exactly the same as No. 49. Beam weighed 58 pounds, broke at 9900 pounds. Check-pieces: A weighed 21 pounds, broke at 3000 pounds; D weighed 19 pounds, broke at 3300 pounds. Efficiency of beam, 78.5 per cent. Mean efficiency of the two beams, 72.8 per cent.

One objection often urged against driving keys is the liability of splitting out the daps between the key-slots. The writer has encountered no such trouble, but on the contrary has been surprised to find that, even in Nos. 28, 29A, and 29B, the keys could be driven up till their edges would be battered up, without in any single case splitting out a dap. The ability to withstand such hard driving is the cause of the superior stiffness of beams with wedges. It will be noticed, on comparing curves for beams, Nos. 41 to 50, that in every case the wedged keys produce a stiffer beam than can be got by the other constructions.

It should further be stated that in the preceding cases none of the sticks failed by shearing out the dap, and that every break began on the tension-side of the beam.

Summary of Tests on Beams with Special Forms of Iron Key.

Number of Beam.	Ratio of Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.	Remarks.
39	16.4	5,700	61.1	No wedges.
40	16.4	7,400	79.3	" "
		Average...	70.2	
41	12.0	7,900	53.6	" "
42	12.0	11,100	70.4	" "
43	12.0	7,800	51.1	" "
44	12.0	10,200	73.2	" "
		Average...	62.1	
45	12.0	11,940	88.1	" "
46	12.0	10,630	67.2	" "
47	12.0	11,130	70.3	" "
		Average...	68.8	
48	12.0	10,900	79.7	With wedges.
49	12.0	9,600	67.2	" "
50	12.0	9,900	78.5	" "
		Average...	72.8	

XI. General Conclusions.

It seems clear from these tests that it is practically impossible to construct a built beam which shall be as strong as a solid beam of equally good material. The writer considers that the results of the tests warrant the conclusion that the keyed or joggled beam is by far the best form, as it surpasses all others in both strength and stiffness. Oak, hickory, hornbeam or dogwood keys will suffice for beams constructed of white pine, spruce, or other soft woods, but iron keys should be used for hard-wood beams. Wedge-shaped driving-keys are necessary in order to secure the most efficient beams, and the keys should be of iron if the maximum stiffness is desired. In the centre of the span a space equal to about one-quarter of the length of the beam should be left free of keys, bolts, etc. It will be noted that whenever less space than this was allowed in the tests the efficiency was low, because the fracture occurred in the fastenings. In particular cases it may be necessary to use some bolts within the space mentioned to prevent the compression side from crippling, but such bolts should be as few in number and of as small a diameter as can possibly be permitted.

If this precaution is observed, and the beams are otherwise properly designed, the writer believes that the following efficiencies, etc., may with safety be used in practice, when the compound beam is composed of two pieces.* These values are lower than those first published, but the efficiencies are subject to such great variations that it seems dangerous to use values much higher than those here advised.

	Efficiencies. Per cent.	Deflections. Per cent.
Solid beam,	100	100
Clark's beam,	65	200
Piped beam,	65	175
Keyed beams:		
Oak keys,	75	125
Flat or square wrought-iron keys,	70	150
Cast-iron keys, as in No. 28,	80	100
Other metal keys tested,	65 to 75	100 to 150

* In fixing the values the factor of safety allowed in the structure should be considered. If very low working-stresses are used it might be safe to increase the above efficiencies 5 per cent., but it is seldom that the writer would venture to do this.

The use of other than keyed beams is not advised by the writer.

These values refer, of course, only to beams built of soft woods, such as white pine, spruce, eastern fir, etc. But if iron keys be used they will apply equally well to beams built up of the hard woods.

PART II.

XII. Tests on Norway Pine Beams.

The tests now to be described were made first, and suggested many of the ideas used in Part I. They do not furnish sufficient data for very accurate determination of efficiencies, but it is thought the results are worth recording, since they permit the reader to judge how far any results are applicable to beams framed up without regard to how the grain runs in the various pieces. Diagram 14 shows curves of all beams whose deflection was measured.

All the pieces were 4.75 inches wide, the depths being either 0.9375, 1.875, or 3.75 inches. In all cases the length was 78 inches, and the clear span 72 inches.

Transverse tests on single sticks, 3.75 inches deep and 4.75 inches wide, loaded in center, gave the following results:

A failed at 4230; B at 4800; C at 5950; D at 6000; and E at 5860 pounds. Average, 5368, say 5370, pounds.

This will be used as a basis for figuring the efficiency. The results will of course be only approximate, but will show the general law as to the efficiencies of the various forms. It was not considered worth while to take weight of pieces into account.

Clark's Beam.

No. 51. Built like Fig. 48. Side-struts of 0.5-inch white pine, secured by 1.5-inch wire nails. Broke at 12,800 pounds.

No. 52. Exactly similar to No. 51. Broke at 12,500 pounds.

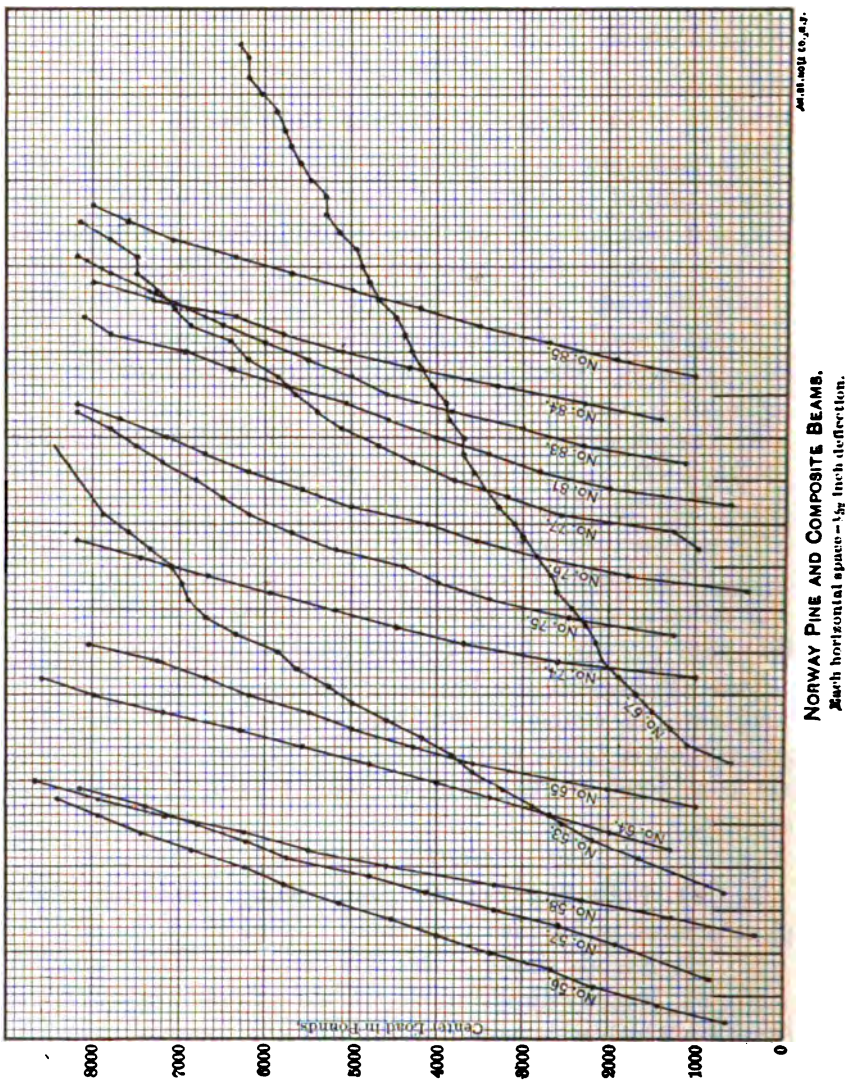
No. 53. Exactly similar to No. 51, except that struts were secured by 2-inch wire nails. Broke at 13,840 pounds.

No. 54. Exactly similar to No. 53. Broke at 13,200 pounds.

No. 55. This beam had two thicknesses of 0.5-inch white-pine struts on each side, the second layer being placed at right angles to the first, and the whole secured with 2.5-inch wire nails. Broke at 16,670 pounds.

No. 56. Built like Fig. 48, but the struts were of selected Norway pine, 0.5-inch thick, secured by 1.5-inch wire nails. Broke at 11,650 pounds.

No. 57. Same as No. 56, except that struts were of 1-inch



Norway pine, secured by 2.5-inch wire nails. Broke at 15,000 pounds.

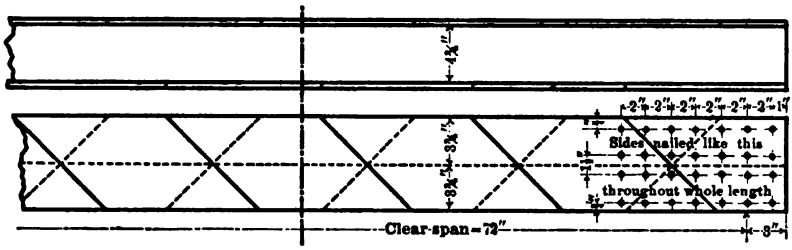
No. 58. Built like Fig. 49. White-pine struts 0.5-inch thick, secured by 1.5-inch wire nails; 8 oak keys 1 by 2 inches, and 12 6-inch steel spikes. Broke at 16,690 pounds.

No. 59. Exactly like No. 58. Broke at 13,200 pounds.

No. 60. Exactly like No. 58. Broke at 14,800 pounds.

No. 61. Four pieces 1.875 inches deep, built up as in Fig.

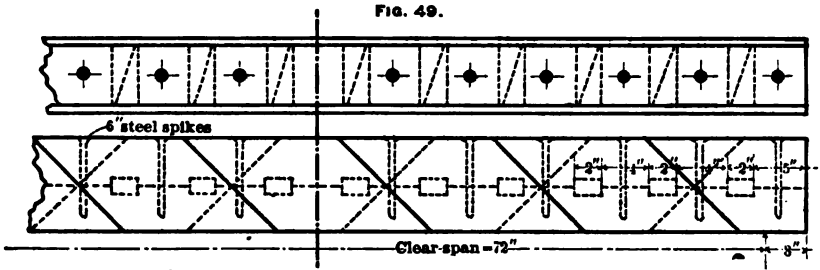
FIG. 48.



CLARK'S BEAM.

50. Struts of 0.5-inch white pine, secured by 2-inch wire nails. Broke at 8140 pounds.

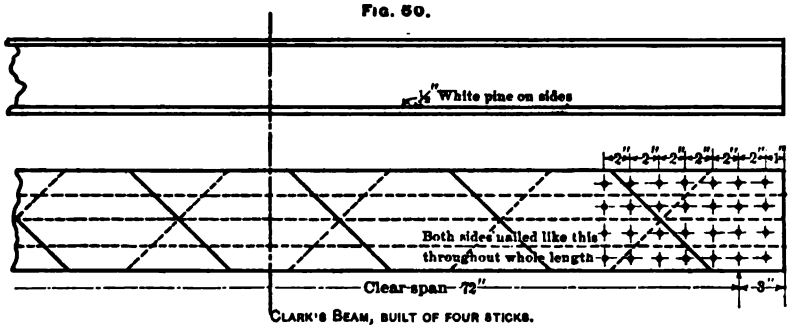
FIG. 49.



CLARK'S BEAM. SIDES OF THIS BEAM WERE NAILED EXACTLY LIKE FIG. 48.

No. 62. Four pieces 0.9375-inch deep, superimposed like Fig. 50, forming a beam 3.75 inches deep. Struts of 0.5-inch

FIG. 50.



CLARK'S BEAM, BUILT OF FOUR STICKS.

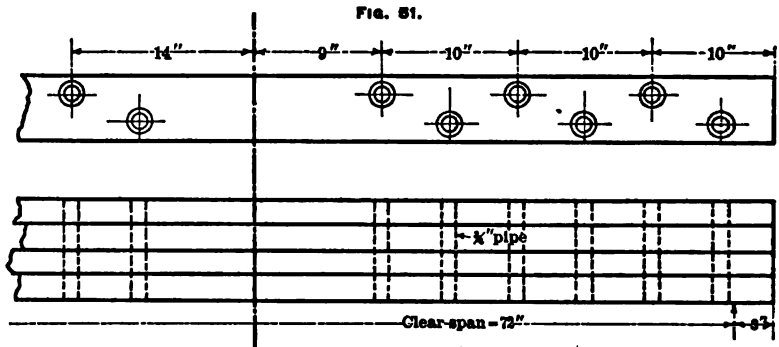
white pine, secured by 1.5-inch wire nails. Broke at 2700 pounds.

Summary of Clark's Beams.

Number of Beam.	Ratio of Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.
51	9.6	12,800	59.7
52	9.6	12,500	58.3
53	9.6	13,840	64.5
54	9.6	13,200	61.5
55	9.6	16,670	77.5
56	9.6	11,650	54.4
57	9.6	15,000	69.9
58	9.6	16,690	77.8
59	9.6	13,200	61.5
60	9.6	14,800	69.0
61	9.6	8,140	38.0
62	19.2	2,700	50.3
		Average.....	61.9

These figures clearly prove the low efficiency of Clark's beam, and likewise indicate that this efficiency will be less for small than for large ratios of span to depth of beam, since in the former case the unit shearing-stresses are increased. Nos.

Fig. 51.



Piped Beam built of four sticks, with ¾" bolts through pipes.

61 and 62 bear out the statement that in general the greater the number of pieces used, the less efficient the beam.

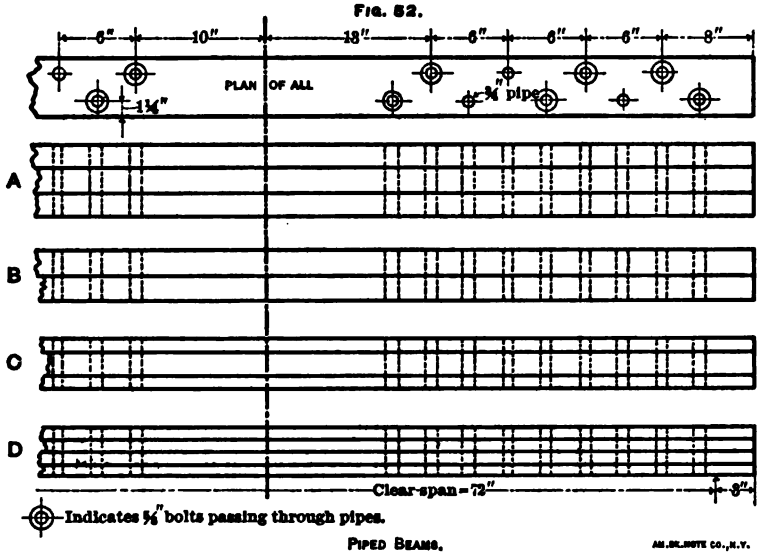
XIII. Beams Secured by Pipes.

No. 63. Four pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, superimposed and secured by 12 pipes and 12 bolts, as in Fig. 51. Broke at 11,300 pounds.

No. 64. Two pieces of Norway pine, each 3.75 inches deep and 4.75 inches wide, secured by 18 pipes and 12 bolts, as in

Fig. 53 (Plan). Tension-side had one very sappy corner, turned toward center of beam. Broke at 10,000 pounds.

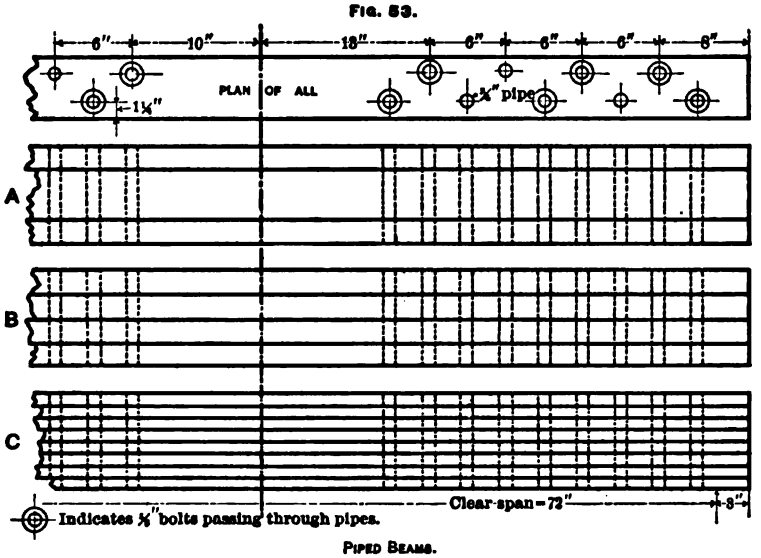
FIG. 52.



PIPED BEAMS.

No. 65. Three pieces. Central piece 8.75 inches deep, and top and bottom pieces each 1.875 inches deep; all 4.75 inches

FIG. 53.



PIPED BEAMS.

wide and secured by 18 pipes and 12 bolts, as in Fig. 53 (A). Broke at 14,800 pounds.

No. 66. Four pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, superimposed and secured by 18 pipes and 12 bolts, as in Fig. 53 (B). Broke at 11,000 pounds.

No. 67. Eight pieces of Norway pine, each 0.9375-inch deep and 4.75 inches wide, superimposed and secured by 18 pipes and 12 bolts, as in Fig. 53 (C). Broke at 10,300 pounds.

No. 68. Three pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, superimposed and secured by 18 pipes and 12 bolts, as in Fig. 52 (A). Broke at 6300 pounds.

No. 69. Two pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, secured by 18 pipes and 12 bolts, as in Fig. 52 (B). Broke at 5000 pounds.

No. 70. Exactly similar to No. 69. Broke at 4540 pounds.

No. 71. Three pieces of Norway pine. Central piece 1.875 inches deep, and top and bottom pieces each 0.9375-inch deep; all 4.75 inches wide. Secured by 18 pipes and 12 bolts, as in Fig. 52 (C). Broke at 4840 pounds.

No. 72. Four pieces of Norway pine, each 0.9375-inch deep and 4.75 inches wide, superimposed and secured by 18 pipes and 12 bolts, as in Fig. 52 (D). Broke at 4800 pounds.

Summary of Beams Secured by Pipes.

Number of Beam.	Ratio of Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.
63	9.6	11,300	52.7
64	9.6	10,000	46.6
65	9.6	14,800	69.0
66	9.6	11,000	51.3
67	9.6	10,300	48.0
68	12.8	6,300	52.2
69	19.2	5,000	93.2
70	19.2	4,540	84.5
		Average.....	62.2

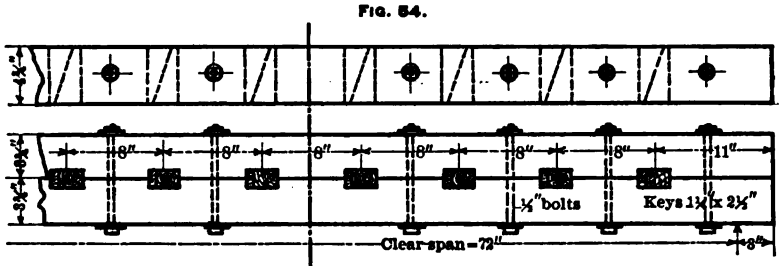
These results confirm, in a rough way, the conclusion that these beams are stronger than Clark's. The efficiencies of the piped beams would have been higher had fewer pipes been used, so as to allow greater spacing. Every fracture took place between two pipes.

XIV. Beams with Hardwood Keys.

No. 73. Two pieces of Norway pine 3.75 inches deep and 4.75 inches wide, secured by 4 0.5-inch bolts and 8 oak keys, as in Fig. 54. Failed at 13,800 pounds. Broke on tension-side.

No. 74. Two pieces of Norway pine 3.75 inches deep and 4.75 inches wide, secured by 12 6-inch steel spikes and 12

Fig. 54.

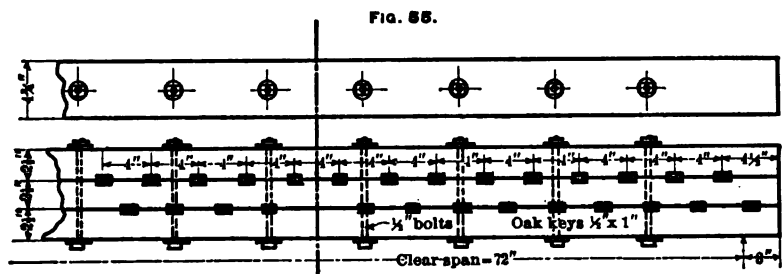


Joggled Beam with Hardwood Keys.

white-ash keys, like Fig. 49, with side-struts omitted. Broke at 12,900 pounds. Failed by shearing the keys.

No. 75. Three pieces of Norway pine, each 2.5 inches deep and 4.75 inches wide, secured by eight 0.5-inch bolts and 36 white-oak keys, as in Fig. 55. Broke at 12,070 pounds. Sheared part of the keys, and broke on tension side.

Fig. 55.



Joggled Beam with Hardwood Keys.

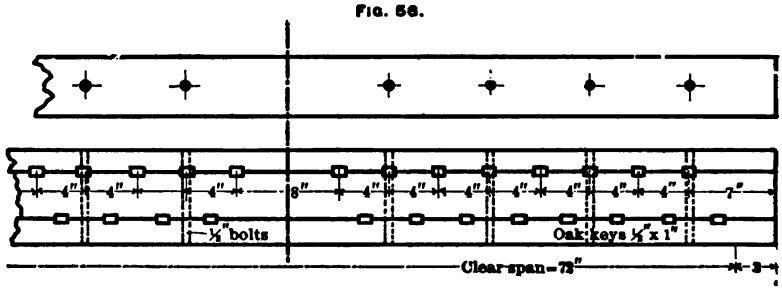
No. 76. Three pieces of Norway pine. Central piece 3.75 inches deep, top and bottom pieces each 1.875 inches deep; all 4.75 inches wide, secured by eight 0.5-inch bolts and 32 white-oak keys, as in Fig. 56. Broke at 13,200 pounds. Sheared part of keys, then broke on tension side.

No. 77. Four pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, secured by 0.5-inch bolts and keys, as in

Fig. 57. Broke at 11,700 pounds. Keys sheared first, then tension side broke.

No. 78. Two pieces of Norway pine, each 1.875 inches deep

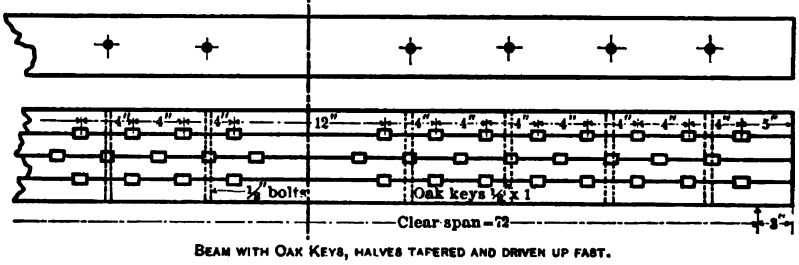
FIG. 56.



BEAM WITH OAK KEYS, HALVES TAPERED AND DRIVEN UP FAST.

and 4.75 inches wide, secured by 0.5-inch bolts and oak keys, as in Fig. 58. Broke at 4670 pounds. Failed on tension side. Keys remained intact.

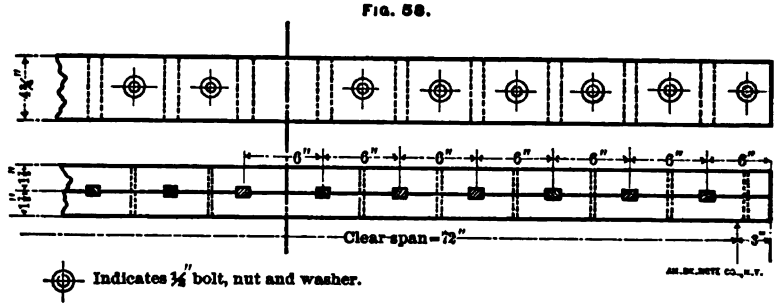
FIG. 57.



BEAM WITH OAK KEYS, HALVES TAPERED AND DRIVEN UP FAST.

These results show clearly the inadequacy of hardwood keys in hardwood beams, particularly for short spans. The efficiencies just given are little, if any, higher than would have

FIG. 58.



been got by merely superimposing the pieces, without any fastenings at all. The keys indented so readily that the pieces acted almost independently of each other. In nearly

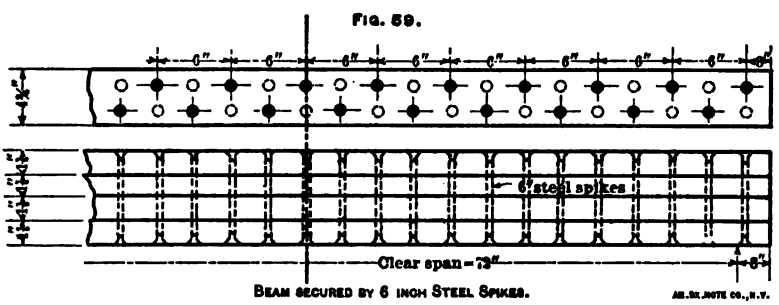
Summary of Norway Pine Beams with Hardwood Keys.

Number of Beam.	Ratio of Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.
73	9.6	13,800	64.3
74	9.6	12,900	60.1
75	9.6	12,070	56.3
76	9.6	13,200	61.5
77	9.6	11,700	54.5
		Average.....	59.3
78	19.2	4,670	87.1

every case part of the keys sheared first, and then the tension side of the beam failed, showing the correctness of the view that a beam with concentrated load fails by longitudinal shearing, instead of by transverse stress, when ratio of span to depth of beam is less than 10.

No. 79. Four pieces of Norway pine, each 1.875 inches deep and 4.75 inches wide, secured by 6-inch steel spikes, as in Fig.

FIG. 59.



59. Failed at 7900 pounds. Efficiency of beam, 36.8 per cent.

No. 80. Exactly the same size of pieces as in No. 79, but merely superimposed, without fastenings of any kind. Broke at 6700 pounds. Bottom piece broke first, then the top, next the piece nearest bottom, and finally the piece nearest the top. Efficiency of beam, 31.2 per cent.

The results of these tests show that nails or spikes are of no value for this kind of work, as they pull out under the stress. Most of the resistance of No. 79 came slowly from the friction between the pieces, as is evident from No. 80.

XV. Composite Beams.

Some writers advocate the use of hardwood on the tension side of a beam. The following beams were therefore built of pine and oak, as specified in each case.

No. 81. Central piece of Norway pine 3.75 inches deep; top piece, Norway pine, and bottom piece of white oak, each 1.875 inches thick; all pieces 4.75 inches wide. Secured by 18 pipes and 12 bolts, as in Fig. 53 (A). Broke at 15,180 pounds. Middle piece failed first.

No. 82. Exactly the same as No. 81, except that both top and bottom pieces were of white oak. Broke at 13,460 pounds. Middle piece failed first.

No. 83. Central piece of Norway pine 3.75 inches deep; bottom piece of Norway pine, and top piece of white oak, each 1.875 inches deep; all 4.75 inches wide. Secured by 0.5-inch bolts and oak keys, like Fig. 56. Failed at 13,980 pounds, by shearing the keys.

No. 84. Exactly like No. 83, except that the oak was put on the bottom instead of on the top. Sheared the keys, then broke in center piece, and next the bottom piece failed, the maximum load being 15,200 pounds.

No. 85. Exactly like No. 83, except that both top and bottom pieces were of white oak. Keys indented badly, then bottom piece gave way all at once. None of the keys sheared. Broke at 13,200 pounds.

No. 86. Three pieces, each 1.875 inches deep and 4.75 inches wide. Bottom piece of white oak, and the other two of Norway pine. Secured by 18 pipes and 12 bolts, as in Fig. 52 (A). Broke at 8460 pounds.

Summary of Composite Beams.

Number of Beam.	Ratio of Span to Depth of Beam.	Breaking Load. Pounds.	Efficiency. Per Cent.
81	9.6	15,180	70.8
82	9.6	13,460	62.8
83	9.6	13,980	65.2
84	9.6	15,200	70.9
85	9.6	13,200	61.5
86	12.8	8,460	70.1
		Average.....	66.9

The results show that the oak certainly increased the efficiency; but it seems to the writer that here, as in the case of beam No. 19, the increase is not sufficient to compensate for the extra cost of procuring and working the oak. If slightly larger pine beams be used, the same results can be obtained at a smaller cost than by using the composite beam. Wooden keys are not strong enough to develop the full strength of the oak; and the writer believes that had iron keys been used the results would have been much more favorable.

Diagram 14 shows the deflection characteristic of these beams. It will be noted that Clark's forms come out almost as good as the others. This is due solely to the fact that, with a small ratio of span to depth, wooden keys are not strong enough to develop the strength of the component pieces in the built beam. No. 78 shows how the efficiency increases as soon as a ratio is reached which permits the use of enough keys to keep the bearing-stress on them within safe limits.

Nos. 63 and 67 confirm the conclusions drawn in Part I., that the greater the number of component pieces, the greater the deflection.

PART III.

XVI. Formulas for Designing Built Beams.

Since the tests show that the keyed beams are superior to the other forms, and such beams are as cheap to build as any other, the following computations will be confined wholly to this form. The beam is supposed to be of rectangular cross-section, and of uniform size from end to end. Let

M = bending moment in inch-pounds at any section, A B C D, Fig. 60.

M' = similar bending moment at section a b c d.

I = moment of inertia of section of beam.

V = total vertical shearing-stress on any cross-section of beam = reaction on support, minus all loads between reaction and section in which shear is considered.

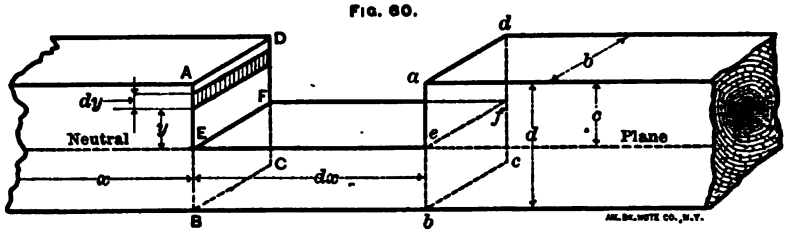
P = load in pounds concentrated in center of beam.

P' = concentrated load at any point in the beam.

w = uniformly distributed load in pounds, per inch length of beam.

W = total uniform load in pounds on beam, = w l.
l = length of span in inches = 12 L.
L = length of span in feet.
d = depth of beam in inches.
b = breadth of beam in inches.
c = least distance in inches from neutral plane to outermost fiber in beam = $\frac{d}{2}$
f = total shearing-stress in pounds in neutral plane of beam carrying load in center.
F = similar stress for beam carrying a uniformly distributed load.
R = safe unit shearing-stress, parallel to grain, per square inch, in pounds.
S = unit tensile or compressive stress in fiber of wood most remote from neutral plane.

FIG. 60.



T = safe unit compressive stress, pounds, across grain (side pressure).
U = safe unit compressive stress, pounds, on end wood.
N = number of keys needed.
s = breadth of one key, inches.
t = thickness of one key, inches.
p = distance center to center of keys, inches.

From formula $M = \frac{SI}{c}$, to be found in any text-book on resistance of materials, it follows that, within the elastic limit,

Unit skin-stress in fibers along $A\ D = \frac{cM}{I}$.

Unit skin-stress in fibers along $a\ d = \frac{cM'}{I}$.

Assuming an element of area bdy (Fig. 60), distant y from neutral plane,

stress on bdy in face A E D F is $\frac{ycMbdy}{cI}$

and stress on bdy in face a e d f is $\frac{ycM'bdy}{cI}$,

hence, stress on the whole face A E D F is

$$\frac{Mb}{I}\int_0^{\frac{d}{2}} ydy$$

and stress on whole face a e d f is $\frac{M'b}{I}\int_0^{\frac{d}{2}} ydy$;

but $M' = M + dM$, hence difference between total stress in the two faces is

$$\left(\frac{M+dM}{I} - \frac{M}{I}\right) b\int_0^{\frac{d}{2}} ydy = \frac{bd^2dM}{8I}.$$

This must be supplied by a resisting shearing-stress in the plane E F f e. Let f' = magnitude of this stress per square unit; then

$$f'bdx = \frac{bd^2dM}{8I}, \text{ hence, } f' = \frac{d^2dM}{8Idx}.$$

But $I = \frac{bd^3}{12}$. Now let Q = reaction on support of beam, then

$$M = Qx - \frac{wx^2}{2} - \text{sum of } P'x,$$

hence $\frac{dM}{dx} = Q - wx - \text{sum of } P'$, but this is equal to the vertical shear V, hence $f' = \frac{12Vd^2}{8bd^3} = \frac{3V}{2bd}$.

But $\frac{V}{bd}$ = mean unit vertical shearing-stress, hence the horizontal unit shearing-stress at any point in the neutral plane is 1.5 times the mean unit of vertical shearing-stress at the same point.

Application to the Case of a Beam Loaded in the Center.

In building up a compound beam the problem is to put in the neutral plane fastenings sufficient to resist the shearing stress just determined. When a beam is loaded in the center, the vertical shear V is everywhere, except directly under the

load, equal to $\frac{P}{2}$, hence the horizontal unit shearing-stress is $\frac{3}{2} \times \frac{P}{2\ bd,}$ and is constant throughout the entire neutral plane. Hence the total stress to be resisted by the keys is, unit stress $\times$ area neutral plane $= bl \times \frac{3P}{4bd} = \frac{3Pl}{4d}$.

Application to Beam Uniformly Loaded.

Reaction on support $= \frac{W}{2}$.

Vertical shear V in any section distant x from support $= \frac{W}{2} - wx$;

and unit horizontal shear in neutral plane at that point $= \frac{3}{2bd}\left(\frac{W}{2} - wx\right)$.

Shear on element of length of beam equal to dx will therefore be $\frac{3}{2bd}\left(\frac{W}{2} - wx\right)bdx$.

Hence, total shear in neutral plane which must be resisted by the keys is:

$$F = \frac{2 \times 3}{2d}\int_0^{\frac{l}{2}}\left[\frac{W}{2} - wx\right]dx = \frac{3}{d}\left[\frac{Wl}{4} - \frac{wl^2}{8}\right]$$

$$= \frac{3}{d}\left[\frac{Wl}{4} - \frac{Wl}{8}\right] = \frac{3\,Wl}{8d}, \text{ since } wl = W.$$

If the same factor of safety be used in each case, a beam which can support a center-load P, can carry a uniformly distributed load of 2P; hence $W = 2P$; hence

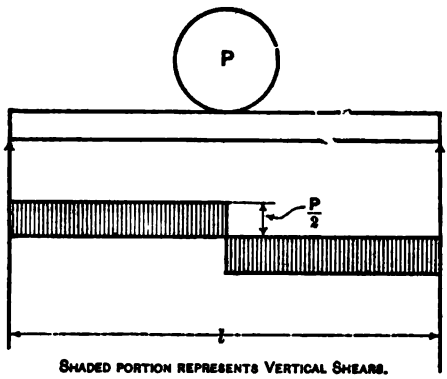
$$\frac{3Pl}{4d} = \frac{3\,Wl}{8d}; \text{ therefore,}$$

in any beam of given dimensions, the total shearing-stress along the neutral plane will be constant, whether the load be concentrated in the center or uniformly distributed, regardless of the length of the beam, if the same factor of safety be used in all cases.

It must be noted, however, that the distribution of the

stresses is not the same in each case. With a center-load, the unit-shear will be constant at every part of the neutral plane, as in Fig. 61, while with a distributed load the shear is a maximum at the support, and decreases uniformly to zero at center of the span, as in Fig. 62.

FIG. 61.

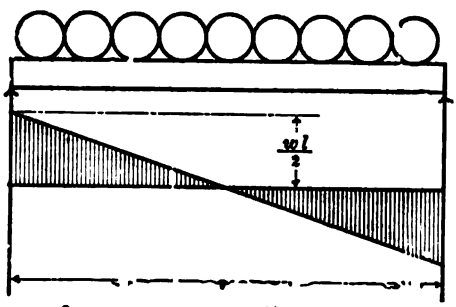


SHADED PORTION REPRESENTS VERTICAL SHEARS.

XVII. ALLOWABLE WORKING-STRESSES IN WOOD.

The writer has adopted as a basis for computation the allowable working-stresses recommended by "The Committee on Strength of Bridge and Trestle Timbers," adopted by the

FIG. 62.



SHADED PORTION REPRESENTS VERTICAL SHEARS.

Association of Railway Superintendents of Bridges and Buildings (Fifth Annual Convention, New Orleans, La., Oct., 1895).* The only change in the values recommended by the committee is the introduction of the compressive strength of hemlock, and change in the compressive stress across grain for oak. The

* *Proceedings* for 1895.

recommended value for oak is 500, while the writer uses 800 for oak keys, for the following reason: The committee had in mind full-sized timber, not always of the best quality or perfectly dry. Keys, however, are always made of the best and driest wood obtainable, and for such material the writer thinks a working-stress of 800 pounds is not excessive.

Table of Safe Working-Stresses, Pounds per Square Inch.

Kind of Timber.	Compression.		Shearing.
	End Wood. U.	Across Grain. T.	With Grain. R.
Factor of Safety	Five.	Four.	Four.
White oak	1,400	500*	200
White pine	1,100	200	100
Southern, long-leaf, or Georgia, yellow pine	1,600	350	150
Douglas, Oregon and Washington yellow fir	1,600	300	150
Northern, or short-leaf, yellow pine, hemlock, spruce, and eastern fir	1,200	.....	100

XVIII. Finding the Number and Spacing of Wooden Keys Needed.

It is evident from the preceding deductions that in a beam centrally loaded the keys should be spaced equidistant along the neutral plane, except that none should be put in the center, where the bending-moment is greatest. The keys in a uniformly loaded beam should be most numerous toward the end of the beams, like rivets in a plate-girder flange. It is impracticable to make the size and spacing of keys conform exactly to the theoretical requirements; hence the writer suggests the following scheme:

In a beam centrally loaded, determine the number of keys needed to resist the shearing-stresses, leave in the middle of the beam a length equal to one-quarter of the span free from fastenings of any kind (unless small bolts), then space the keys at equal distances throughout the remaining part of the beam. The writer believes that, when practicable, it is better to use a number of small keys rather than a few large ones, since the beam is less weakened by the key-slots, and in proportion as

* 800 for oak keys.

the keys are more numerous their action approaches more nearly to the conditions found in a solid beam.

Since oak is practically the only material that is generally available for keys, the formulas will assume oak, but can, of course, be used for any material by substituting the proper constants.

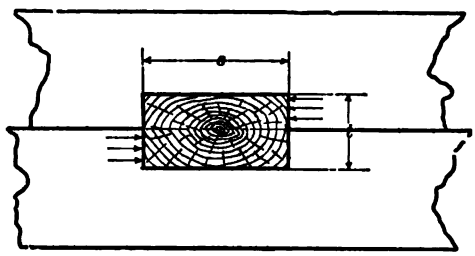
If the key is to be equally strong against shearing and side-pressure, then (Fig. 63):

$$T \times \frac{t}{2} = sR; \text{ hence, for oak}$$

$$\frac{s}{t} = \frac{T}{2R} = \frac{800}{400} = 2.$$

Hence the width of an oak key must be twice its thickness.

FIG. 63.



To Find the Number and Spacing of Keys for a Beam Loaded in the Center.

Total stress to be resisted $= \frac{3Pl}{4d}$*.

Resistance of one key $= \frac{btT}{2}$.

If N = number of keys needed, then $N = \frac{3Pl}{2bdtT} = \frac{18PL}{800bdt}$ $= \frac{0.0225PL}{bdt}$.

Spacing of keys. Length of beam available for keys is $\frac{3l}{4}$; hence,† spacing of keys (inches):

* When working out N, the writer usually takes P as the full theoretical load of a solid beam, as the greater the number of keys the stiffer the built beam.

† It has not been thought worth while to complicate the formulas by taking into account the fact that the number of spaces is one less than number of keys.

$$p = \frac{8l}{4} \times \frac{2bdtT}{8Pl} = \frac{bdtT}{2P} = \frac{400bdt}{P} \text{ for oak keys.}$$

It will be convenient at times to assume the spacing, and find the size of key to fit, hence for that case

$$t = \frac{Pp}{400bd}$$

Example.—Find number and spacing of keys for a compound beam made of two sticks of white pine 8 inches wide and 10 inches deep, span, 24 feet. The allowable extreme fiber-stress for white pine (ordinary half green wood) is 700 pounds; hence a solid beam 8 by 20 inches of the given span will support a concentrated load of 5185 pounds. Assume that we use oak keys in which $t = 1$ inch, and $s = 2$ inches, then

$$N = \frac{0.0225 \times 5185 \times 24}{8 \times 20 \times 1} = 17.5, \text{ say 18 keys.}$$

$$\text{And } p = \frac{400 \times 8 \times 20}{5185} = 12.3 \text{ inches.}$$

To Find Number and Spacing of Keys for Uniformly Loaded Beams.

Total shear in neutral plane $= \frac{3Wl}{8d}$

as before, resistance of one key $= \frac{btT}{2}$,

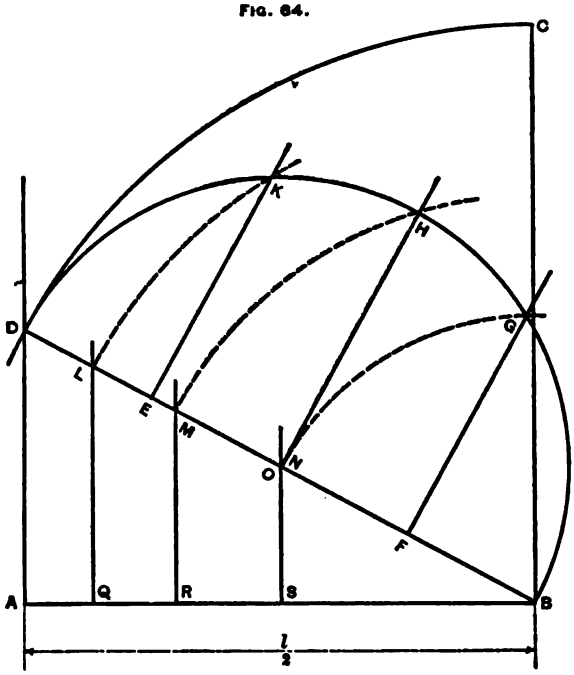
hence $N = \frac{3Wl}{8d} \times \frac{2}{btT} = \frac{3Wl}{4bdtT}$, or for oak keys

$$N = \frac{3Wl}{4 \times 800bdt} = \frac{3 \times 12WL}{4 \times 800bdt} = \frac{0.01125WL}{bdt}$$

The spacing cannot be worked out as easily as in the last case. Since the shearing-forces are distributed as in Fig. 62, we must so space the keys that the perpendiculars from the center of each key will divide up the shaded triangle into parts of equal area; every key will then do exactly the same duty. Fig. 64 shows an easy graphical solution of the problem. Lay off line AB to any convenient scale to represent half the span. Find N from the formula, erect perpendicular at B, and lay out

distance BC any convenient length greater than AB, and so that it will divide out easily into $\frac{N}{2}$ parts. With center B, and radius BC, strike an arc CD, intersecting at D a perpendicular from A. Divide DB into $\frac{N}{2}$ equal parts, say at E, O, and F. At these points erect perpendiculars to line DB, till they intersect at G, H, and K, a semicircle described on DB as a diameter. With center B, and radii BG, BH, and BK, strike out the points L, M, and N; from these points drop perpendiculars

FIG. 64.



on AB, and locate Q, R, and S, then will A, Q, R, and S be the proper position for keys along the half length of the beam, the point A being at the wall or support.*

Example.—Assume same size of beam and keys as in the preceding case, but carrying a load uniformly distributed. The allowable load would then be, considering flexure only, $2 \times 5185 = 10{,}370$ pounds; hence,

$$N = \frac{0.01125 \times 10{,}370 \times 24}{8 \times 20 \times 1} = 17.5, \text{ or say 18 keys, as before,}$$

* *Cf.* Angel's *Practical Plane Geometry and Projection*, p. 46.

since the total longitudinal shear is the same. Solving for spacing by means of the diagram, and putting first key at the wall,* the pitches of the 9 keys starting from the wall will be approximately 8.5, 9, 9.5, 10.5, 11.5, 13, 15.5 and 19.5 inches. In the following section it will be shown that the theoretical minimum safe spacing for oak keys is

$p = \left(\frac{400 + 2R}{R}\right) t$, where R = safe shearing-stress parallel to grain of material of which beam is made. For white pine R = 100, hence

$p = \left(\frac{400 + 200}{100}\right) \times 1 = 6$ inches. To this we should add about 2 inches to compensate for extra stress due to driving up keys, hence the minimum actual working space or pitch would be about 8 inches, so that the values above found are safe.

The writer does not take into account the resistance to shearing contributed by the bolts. All the experiments, and particularly those on Nos. 79 and 80, show that, apart from the friction between the faces of the component sticks, bolts contribute little or nothing in the way of resistance to shearing. The frictional resistance is of value, but is too uncertain a quantity to take into account when computing the number of keys wanted.

Minimum Safe Spacing of Hardwood Keys.

The resistance of the beam to shearing between the key-slots should be equal to the strength of the keys, hence (Fig. 65)

$$R \times \text{distance } a\text{—}b = \frac{t}{2} \times T,$$

$$\text{or } (p - s)\, R = \frac{tT}{2}; \text{ but } s = 2t; \text{ hence}$$

$$(p - 2t)\, R = \frac{tT}{2}, \text{ or } p = \frac{\left(\frac{T}{2} + 2R\right)t}{R} = \frac{(400 + 2R)}{R} t$$

when oak keys are used, which is the minimum safe spacing allowable for the keys. To provide for the extra pressure

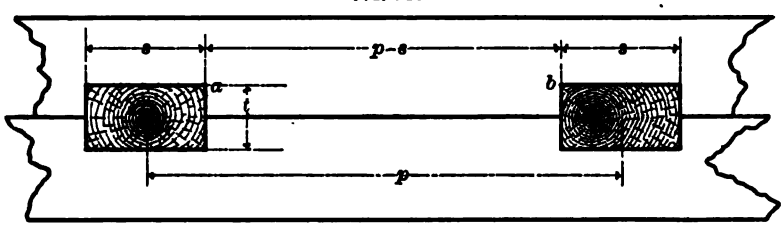
* Distance from this key to end of beam would of course have to be not less than 8 inches. If this distance is not available, owing to short length allowed for bearing on the wall, the only recourse is to move all the keys nearer center of beam, till the 8 inches are obtained.

caused by driving up the keys, the writer would increase these values 25 to 40 per cent. in practice. If the spacing worked out by the formula $p = \frac{400bdt}{P}$, and by using diagram, Fig. 65, should be less than that given by the formula just deduced, the only recourse is to use iron keys.

XIX. Determination of the Number and Spacing of Iron Keys Needed.

The maximum pressure allowable on an iron key is dependent only on the crushing strength of the end-wood bear-

Fig. 65.



ing against the key. Hence it is clear that the preceding formulas hold good if we substitute U, the safe compressive stress on the end-wood, in place of T. Doing this, we obtain the following results:

Iron Keys. Beam Loaded in Center.

$$\left(\text{Formula, } N = \frac{3Pl}{2\,bdtU}\right)$$

Material of Beam.	Number of Keys.*	Spacing of Keys.
White oak	$N = \frac{3Pl}{2800\,bdt}$	$p = \frac{700\,bdt}{P}$
White pine	$N = \frac{3Pl}{2200\,bdt}$	$p = \frac{550\,bdt}{P}$
Southern, long-leaf, or Georgia, yellow pine	$N = \frac{3Pl}{3200\,bdt}$	$p = \frac{800\,bdt}{P}$
Northern, or short-leaf, yellow pine, hemlock, spruce, and eastern fir	$N = \frac{Pl}{800\,bdt}$	$p = \frac{600\,bdt}{P}$

* Note that l is span in inches.

Iron Keys. Beam Uniformly Loaded.

$$\left(\text{Formula, } N = \frac{3Wl}{4bdtU}\right)$$

Material of Beam.	Number of Keys.*
White oak	$N = \frac{3Wl}{5600bdt}$
White pine	$N = \frac{3Wl}{4400bdt}$
Southern, long-leaf, or Georgia, yellow pine	$N = \frac{3Wl}{6400bdt}$
Northern, or short-leaf, yellow pine, hemlock, spruce, and eastern fir	$N = \frac{Wl}{1600bdt}$

Spacing must be worked out by the graphical method, Fig. 64.

Minimum Safe Spacing of Iron Keys.

Substituting U in place of T in formula worked out for wooden keys, we have $(p - s)R = \frac{tU}{2}$. Assuming, as before, that $s = 2t$, since this proportion gives a key which has sufficient resistance to turning around in the slot, $(p - 2t)R = \frac{tU}{2}$,

hence $p = \frac{\left(\frac{U}{2} + 2R\right)t}{R}$,

which reduces to the following values,

For white oak, $p = 5.5t$.

For white pine, $p = 7.5t$.

For southern, long-leaf, or Georgia pine, $p = 7.3t$.

For hemlock, northern or short-leaf yellow pine, spruce, and eastern fir, $p = 8t$.

The writer advises that these values be increased 25 per cent. in practice, to provide for the extra stresses due to driving the keys into place.

PART IV.

XX. Modulus of Rupture of White Pine.

While the primary object of this investigation was to determine the efficiency of built beams, it is thought that the

* Note that l = span in inches.

Moduli of Rupture of White Pine.

d.	b.	l.	W.	P.	$P+\frac{W}{2}$	S.
2.75	4.75	66	17	2700	2708	7,460
"	"	"	18	2600	2609	7,190
"	"	"	16	1880	1888	5,200
"	"	"	16	1980	1988	5,480
"	"	"	18	2400	2409	6,640
"	"	"	18	3500	3509	9,670
"	"	"	15	1830	1838	5,070
"	"	"	17	2150	2158	5,950
"	"	"	16	2150	2158	5,950
"	"	"	17	2640	2648	7,300
"	"	"	16	2270	2278	6,280
"	"	"	14	2150	2157	5,950
"	"	"	16	1780	1788	4,930
"	"	"	18	2200	2209	6,090
"	"	"	17	2730	2739	7,560
"	"	"	17	3100	3109	8,570
"	"	"	15	1920	1928	5,310
"	"	"	16	2190	2198	6,060
"	"	"	16	3000	3008	8,290
"	"	"	16	3190	3198	8,810
"	"	"	14	2100	2107	5,810
"	"	"	16	2330	2338	6,440
"	"	"	13	2870	2877	7,930
"	"	"	13	3440	3446	9,500
"	"	"	13	2930	2937	8,090
"	"	"	15	2080	2087	5,750
"	"	"	13	2020	2027	5,590
"	"	"	14	2520	2527	6,960
"	"	"	14	2100	2107	5,810
"	"	"	13	2540	2547	7,020
"	"	"	16	1860	1868	5,150
"	"	"	14	2460	2467	6,800
"	"	"	13	2060	2067	5,700
"	"	"	12	2540	2546	7,020
"	"	"	14	2930	2937	8,090
"	"	"	14	2830	2837	7,820
"	"	"	15	3100	3108	8,570
"	"	"	17	3480	3489	9,620
"	"	"	13	2150	2157	5,950
"	"	"	14	2970	2977	8,210
"	"	"	12	2450	2456	6,770
"	"	"	12	2600	2606	7,180
"	"	"	14	2530	2537	6,990
"	"	"	14	2920	2927	8,070
"	"	"	14	2530	2537	6,990
"	"	"	14	2170	2177	6,000
"	"	"	17	2100	2109	5,810
"	"	"	14	2400	2407	5,270
"	"	"	13	2690	2697	7,430
"	"	"	14	2860	2867	7,900
"	"	"	15	2800	2807	7,740
"	"	"	14	3120	3127	8,620
"	"	"	16	2350	2358	6,500
"	"	"	16	2620	2628	7,240
"	"	"	15	3050	3058	6,690
"	"	"	15	1800	1808	4,980
"	"	"	15	2200	2208	5,550
"	"	"	14	3050	3057	8,430
"	"	"	16	2700	2708	7,460

Moduli of Rupture of White Pine—Continued.

d.	b.	l.	W.	P.	$P + \frac{W}{2}$	S.
2.75	4.75	66	14	2100	2107	5,810
"	"	"	14	2260	2267	6,250
"	"	"	14	2310	2317	6,390
"	"	"	14	2240	2247	6,190
"	"	"	16	2080	2088	5,760
"	"	"	16	2120	2128	5,870
"	"	"	16	1830	1838	5,070
"	"	"	16	2650	2658	7,330
3.75	4.5	"	20	2960	2970	4,650
2.375	5.0	72	13	1600	1607	6,150
"	"	"	13	1600	1607	6,150
1.375	4.75	66	8	600	604	6,660
"	"	"	8	520	524	5,780
"	"	"	7	600	604	6,660
"	"	"	7	750	754	8,310
2.75	4.50	"	15	3500	3508	10,250
"	"	"	14	2150	2157	6,280
"	"	"	14	3200	3207	9,330
"	"	"	13	2670	2676	7,790
"	"	"	14	2880	2887	8,400
"	"	90	17	2360	2368	6,890
"	"	"	17	2300	2309	6,720
2.375	4.0	66	9	1260	1265	5,550
"	"	"	9	1270	1275	5,600
"	"	"	9	1270	1275	5,600
"	"	"	9	1210	1215	5,330
"	"	"	9	1150	1155	5,070
"	"	"	9	1200	1205	5,290
"	"	"	9	1140	1145	5,020
"	"	"	9	1260	1265	5,550
2.75	4.5	108	22	2600	2611	7,600
2.375	"	"	18	1420	1429	4,810
"	"	"	17	1480	1487	5,020
2.375	4.5	"	18	620	629	4,020
"	"	"	18	730	739	4,720
"	4.25	"	18	720	729	4,930
"	"	"	16	790	798	5,390
"	4.5	"	18	870	879	5,610
"	"	"	18	900	909	5,800
3.75	4.5	90	23	3680	3692	7,880
"	"	"	22	3700	3711	7,920
"	"	"	25	4080	4092	8,730
"	"	"	22	3800	3811	8,130
"	"	"	22	3230	3241	6,920
"	"	"	20	3670	3680	7,850
"	"	"	22	3370	3381	7,220
"	"	"	20	3400	3410	7,280
"	"	"	21	4000	4010	8,560
"	"	"	21	3910	3921	8,370
"	"	"	21	3450	3460	7,380
"	"	"	21	3385	3396	7,250
"	"	"	21	3600	3610	7,700
"	"	"	20	3550	3560	7,600
"	"	"	21	3000	3011	6,420
"	"	"	19	3300	3310	7,060

Average from 114 tests..... 6,760
Maximum from 114 tests..... 10,250
Minimum from 114 tests..... 4,020

modulus of rupture computed from results of the tests on the 114 white-pine check-pieces, etc., is worth recording. This modulus is computed from the formula $S = \frac{3l\left(P + \frac{W}{2}\right)}{2bd^2}$, in which

S = modulus of rupture.
P = load in pounds in center which broke the beam.
W = weight of beam in pounds.
l = span in inches.
b = breadth of beam in inches.
d = depth of beam in inches.

When it is considered that all this pine was number one, clear, and about equally dry, the wide difference in the results merely serves to confirm the correctness of the modern engineering view that the working-stresses allowed in timber of the quality used in structures have, until very recently, been too high.

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DISCUSSIONS.

The Cyanide Process.

Discussion of the Papers of Prof. S. B. Christy, Mr. H. Van F. Furman and Mr. George A. Packard. (See vol. xxvi., pages 709, 721 and 735.)

(Chicago Meeting, February, 1897.)

E. B. WILSON, Salem, Va. (communication to the Secretary): Prof. Christy's paper is, beyond doubt, the most valuable contribution hitherto made by an impartial investigator to the literature of the cyanide process; and the Institute is to be congratulated upon its possession of a member willing to labor so greatly, and to communicate the results of his labors so freely, for the benefit of his professional colleagues.

In offering some criticisms upon Prof. Christy's interpretations and conclusions, I do not question the accuracy of his work, or the facts which it establishes, but only the application of these facts to the theory (and incidentally to the practical operation) of the process.

I start with the proposition, which I think has been proved by experiment, that in the solution of gold by means of alkaline cyanides the various reactions are determined, as to their order and intensity, by the relative position of the elements concerned in the electro-chemical series, or series of voltaic tension. While this theory, originally propounded by Berzelius as a law governing all chemical reactions, has been questioned (and, perhaps, in its universal and unqualified application, disproved) on the evidence of contradictory observations, I think the results reported by Gore and others show it to be a safe guide under the conditions here in question.

From the voltaic-tension series, as given by Gore, I quote the following elements, as those especially involved in this inquiry, prefixing to each the number indicating its position in the series, in the order from electro-positive to electro-negative:

3. Potassium.	27. Silver.
4. Sodium.	28. Antimony.
8. Calcium.	29. Tellurium.
9. Magnesium.	31. Gold.
12. Manganese.	37. Carbon.
13. Zinc.	39. Nitrogen.
15. Iron.	40. Arsenic.
20. Lead.	43. Sulphur.
24. Copper.	45. Bromine.
25. Hydrogen.	46. Chlorine.

47. Oxygen.

According to the well-known principles of the electro-chemical theory, the wider apart any two elements in this series, the greater the electro-motive force of the current they would produce as a voltaic couple; the greater their affinity, or tendency to unite; the greater the amount of heat they liberate in uniting; and conversely, the greater the amount of heat (or equivalent electrical energy) required to decompose the compound formed by their union.

Oxygen, being the most electro-negative element in the voltaic series, is the one which liberates most heat by its union with other elements, and consequently the one most effective in decomposing their compounds.

It follows also from the theory, that in solution involving oxidation, those elements are most readily dissolved which are furthest from oxygen in the electro-chemical series, and evolve by their oxidation the most heat. And it may be asserted, as a corollary proposition, that in cases, both of chemical union and of chemical decomposition, those changes first occur to which there is the least resistance, as the resultant of the relative affinities of the elements present, as measured by their distance apart in the electro-chemical series. In voltaic cells, each metal of the couple is corroded, and evolves heat; but the highest electro-motive force is obtained by coupling two elements widest apart in the series; that is, other things being equal, the most easily corroded metal, coupled with the one least easily corroded, yields the most intense current. Potassium and gold would thus form a stronger voltaic couple than could be formed by either of them with zinc, iron, lead or copper, or by either of these with any other. This may ac-

count, in some measure, for the stability of the auro-potassic cyanide.

But, in my opinion, the action of oxygen is necessary to develop the molecular energy required to effect the reactions resulting, under the conditions of this process, in the union of potassium and gold. Prof. Christy aptly compares the part of oxygen in this case with that which it plays in the circulation of the blood of men and animals. At all events, his thorough investigations confirm the results of my own experiment and practice, in showing that the presence of oxygen, or an equivalent agent, is necessary to the solution of gold by this process. Whether either chlorine or bromine does, in fact, perform directly the work of oxygen in the case under consideration, will be considered below. For the present, I assume that oxygen is required.

Since the development of oxygen by the decomposition of the water present would require the expenditure of more energy than the furnishing of it from the atmosphere or by other artificial means, the process would proceed more slowly in the former case.

The Pelatan-Clerici method obtains oxygen in two ways, namely, by the decomposing action of the electric current upon water, and by the exposure of the charge, through agitation, to the oxygen of the air. This permits the employment of a much weaker cyanide-solution.

Mr. Kendall* furnishes oxygen by the use of sodium dioxide. His method, like that of Pelatan-Clerici, requires careful preliminary determination of the amount of oxygen required.

The representatives of the MacArthur-Forrest patented process, while claiming, as Prof. Christy has pointed out, that the action of oxygen was not necessary, ignorantly included it, nevertheless, in the practical operations of percolation and drainage which they prescribed. For not only would the material which they propose to treat be, in their process, at all times exposed to the atmosphere, which contains free oxygen, but the water with which they dilute their solvent solution always carries, in addition to the oxygen which is one of its constituents, a considerable amount of oxygen in solution (by aeration). To these circumstances may undoubtedly be at-

* U. S. patent, No. 538,522, Apr. 30, 1895.

tributed the effective operation of their process when neither agitation nor extraneous artificial means of supplying oxygen are employed.

Since the double salt, $KAuCy_2$, is very stable, and acid does not readily destroy it, I am inclined to believe that acid and the salts of base metals do not cause in this process the extensive loss of cyanogen which has been attributed to them, but rather that this loss is due to other impurities present in the solution at the moment of the formation of the double cyanide of K and Au. I am aware that this view may be controverted on the ground that KCy is readily decomposed by dilute acids. But it may be replied that this is not the case in the presence of gold—a phenomenon which I ascribe to voltaic action. In this connection also the fact may be mentioned that if the amount of lime necessary for the mineralization of the ore be determined, it seems to make no difference in the result whether this lime be added before the lixiviation or at the time of the latter; the amount of cyanogen consumed being about the same in either case. This proposition, I am aware, contradicts the statement of Mr. Furman, in his paper on "Laboratory Tests," etc. (page 724), to the effect that if an ore be acid "the result will be decomposition of potassium cyanide unless this acidity is destroyed before the cyanide-solution is added." In support of my view I would refer to the experiments described below.

These experiments show also that a 0.1-per-cent. cyanide-solution may dissolve the gold as thoroughly as a 1-per-cent. solution. If the latter were used in a given case, we might expect the loss of cyanide to be ten times as great, depending, however, on the amount of impurities in the ore having affinity for the cyanogen set free.

Strong cyanide-solutions have shown, when applied to acid ores, considerable loss of cyanide, but at the same time a high extraction of gold—higher in a given period than when the ore had been previously neutralized. However, when impurities in the solution are a minimum or entirely absent, it will be found that the stronger the solution, the greater the loss of cyanide.

Accepting Prof. Christy's conclusion (as opposed to the view of Mr. MacArthur) that free oxygen is necessary to the usual reaction effecting the solution of gold in cyanide-solutions, I must

venture to doubt his opinion that bromine is capable of doing the same work as oxygen in this reaction. (See his foot-note on page 741, criticizing the formula suggested by Mr. Mulholland.) I agree with him that strong solutions of bromine and potassium cyanide should be avoided, as occasioning a loss of cyanogen; but I believe that the formation of paracyanogen, to which he refers, requires the liberation, in some way, of oxygen.

The order, in the voltaic series already given, of the elements K (3), C (37), N (39) and Br (45) would indicate that bromine has stronger affinity for potassium than either carbon or nitrogen taken separately; but we have here to deal with the compound negative radical cyanogen (CN), which closely resembles bromine in many respects, and is more volatile, assuming the liquid state under the pressure of three atmospheres, while bromine is liquid at ordinary temperature and pressure. We are justified in the inference that if the compounds of cyanogen require oxygen for the reaction here under consideration, bromine, which stands so near cyanogen, will not effect the reaction without oxygen.

The question is, Does KCy require the presence of oxygen for the formation of the double cyanide with gold?

Mr. W. Bettle has reported* experiments in which gold was dissolved from tailings containing basic ferric sulphate in the absence of even traces of free oxygen, the active agent being cyanogen. The formula which he proposes for the reaction is:

$$4KCy + K_6Fe_2Cy_{12} + 2Au = 2KAuCy_2 + 2K_4FeCy_6.$$

It will be noticed that, according to this explanation, the ferricyanide is reduced to a ferrocyanide, which calls for an oxidizing agent.

Mr. Bettle says in another communication:†

"While potassic ferric cyanide is capable of decomposing water in the presence of an oxidizable material, I maintain that gold in ordinary solutions is not an oxidizable material, and, therefore, for the dissolution of gold cyanogen must be placed in atomic contact with this element in a cyanide solution. . . . . To argue otherwise is to assume that aurous or auric oxide is formed by the oxygen liberated, and one of these oxides dissolves cyanide to form the auro, or auric cyanide of the metal."

* *South African Mining Journal*, May 11, 1895.

† *Ibid.*, May 25, 1895.

Prof. J. C. S. Wells, of Columbia University, has declared* that zinc-potassic cyanide is a solvent for gold, but not when free potassium cyanide is present; and he adds that since in the practice of the cyanide process the solutions do always contain free potassic cyanide, the solvent power of the double zinc-salt is not available.

Prof. Christy and Mr. Bettle require the presence of free cyanogen to effect the reaction in the absence of oxygen, while I think, with Prof. Wells, that the reaction will not take place without oxygen.

Mr. Clannel, in a paper on the Sulman process,† has described certain experiments which throw light upon this subject. He placed in three separate beakers the following solutions: (1) a dilute solution of KCy; (2) a dilute solution of BrCy, and (3) a mixture of the two solutions, and covered each with a watch-glass, on which was suspended a drop of nitrate of silver. The drop over No. 1 gradually became turbid from the formation of cyanide of silver; the drop over No. 2 remained unchanged; the drop over No. 3 rapidly became white, showing that hydrocyanic acid was rapidly liberated from the solution below. This requires hydrogen, and the only source of hydrogen under the circumstances is the decomposition of water, which must liberate, either as free oxygen or as peroxide of hydrogen, the oxygen necessary to the reaction. Mr. Clannel proposes the following equation, based on the fact that HCy is a product of the decomposition:

$$5KCy + 2BrCy + 2Au + H_2O = 2KAuCy_2 + 2KBr + KCyO + 2HCy.$$

If his explanation be correct, the active agent is not cyanogen bromide, but potassium cyanate, which quickens the reaction when bromo-cyanide solutions are employed.

Before seeing Mr. Clannel's article on the Sulman cyanogen-bromide process I had formulated the following reactions, which I considered as occurring, not simultaneously, but in close succession:

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* *Engineering and Mining Journal*, December 14, 1895, p. 559.

† *South African Mining Journal*, May 25, 1895.

I. $3KCy + Au + BrCy + O + H_2O = KAuCy_2 + BrCy,Cy + 2KOH.$

II. $2KOH + BrCy,Cy + Au = KAuCy_2 + KBr + O + H_2O.$

The first of these equations represents the supposition that Cy has more affinity than Br for K. It is evident, however, that any potassic hydrate formed will not long remain unaffected in such a solution, and hence the reaction expressed in the second equation quickly follows, supplying the oxygen necessary for the solution of the gold as an element in the double cyanide. The above equations leave free oxygen as a result, and, if the theory here advanced be tenable, we may expect more cyanogen to be liberated the more oxygen is present. With this result the consumption of cyanide in the process stands connected, and if base metal salts or other impurities are present in abundance the loss will be greater from that cause. It is my impression that too much stress has been laid upon metallic and not enough upon earthy salts in this connection, and I hope to be able to give, at some future time, experimental proof that, by paying due regard to the latter, an increased extraction of gold, coupled with a saving of cyanide, can be realized in practice.

The experiments of my own, to which reference has been made above, were made upon several portions of the same ore-sample at the same time, and occupied the same period of treatment—namely, 15 hours. The results were as follows:

	No. of Experiment.		
	1.	2.	3.
Percentage of KCy in solution before use,	1.0809	0.5404	0.1081
Percentage of KCy in solution after use,	0.5145	0.0837	0.0042
Consumption of KCy by difference,	0.5664	0.4567	0.1039
Extraction in milligrammes, gold,	0.5	2.3	2.2
" " " silver,	72.45	138.4	103.3
Percentage of extraction of total gold and silver,	73.0	86.7	79.0

It is evident that the proper strength of the solution for this particular ore would be between that of experiment No. 2 and that of No. 3, since in the latter case nearly all the cyanide was

consumed, and the amount was shown to be too small for a maximum extraction. It will be noticed that the consumption of cyanide did not influence the percentage of extraction in this case.

The Electrical Precipitation of Gold from Auro-Potassic Cyanide.—Of the three conditions, none of which Prof. Christy says (p. 757) can be secured in precipitating gold electrically from a cyanide ore-extraction solution, two are not essential to the process. The conditions necessary for successful electro-chemical action are: (1) a liquid; (2) a definite chemical compound in the liquid; and (3) that the liquid compound be a conductor of electricity.

These conditions do not necessitate either Prof. Christy's No. 1, a soluble anode of the metal to be deposited, or his No. 2, concentrated solutions of the double cyanide of gold and potassium. As to his No. 3, an electrolyte of constant composition, whereby the proper working conditions may be constantly maintained, it is not, as he thinks, impracticable. The electrolyte is a solution of potassium cyanide. It may be changed chemically by the passage of the current, but in any case the potassium and cyanide remain in some form in the solution. It is true that the electrolyte may offer a high specific resistance; but it is more than likely that impurities in the liquor assist in a measure the passage of the current, or they may be introduced artificially to assist it, as, for instance, in the Pelatan-Clerici process, in which about 1 per cent. of common salt (NaCl) is placed in the bath, for no other purpose than to give the electrolyte a sufficient and practically constant conductivity during the whole operation. If other impurities in the liquor assist the passage of the current, so much the better; if they retard it, their effect is not noticeable.

With such a feeble current as is generated in the zinc-boxes the strength of the cyanide-solutions is of considerable moment, but when electricity is applied to weak cyanide-solutions, the strength of the solution is of very little moment. Prof. Christy's comparison of electro-plating or gilding with the electrical precipitation of gold in this case is not well taken. In the first instance, we want an even deposit in a given time; in the second, we want the gold irrespective of uniformity of deposit.

Professor Christy says with reason that the two processes are different; but all the differences and difficulties which he points out, such as absence of a gold anode, weakness of the solution, formation of aurocyanhydric acid, or imperfect division of the double cyanide into its constituents, may be, as he acknowledges, avoided by the use of currents with sufficient intensity and quantity of electricity. He remarks (on the last line of p. 757) that nobody would think of extracting, from such weak solutions, any other metal than gold by electricity. His reason for this opinion is apparently the high cost of the current required. At least, if the difficulty of maintaining a practically constant electrolyte be overcome, the question of cost seems to be the only one which can justify his conclusion. Let us see what is the weight of this consideration.

To decompose the gold salts and cause this metal to be deposited, a theoretical voltage of about 0.4 volt is required, without a gold anode. Let us assume that in practice it is 1.5 volts, a figure which should satisfy the most conservative. A current of 1.5 volts and of 500 amperes represents one horse-power, which costs, say, 40 cents per twenty-four hours; this estimate being likewise liberal.

Such a current, in a solution of gold employed for gilding, where the whole of the amperage is used to precipitate the gold, would precipitate more than $1000 in gold per hour, and more than $24,000 in twenty-four hours. The cost of the electric energy consumed would therefore not reach $\frac{1}{60.000}$ of the value of the gold. If, in order to exhaust the weak solutions and overcome the difficulties pointed out by Professor Christy, the electric energy and quantity should be increased a hundred-fold, the expense would be $\frac{1}{600}$ of the value of the gold; or if it should be increased 1000 times, it would be but $\frac{1}{60}$ of the value of the gold, which would still not be a prohibitory condition. In fact, when we calculate upon the generation of electricity by hydraulic power we can almost reduce the running cost to that of oil and waste.

The action at the anode in solving the double salt $AuKCy_2$ would be first upon Au, then upon K, as gold is less positive than potassium; and the order of deposition would be the same. The cyanide in part reunites with K forming KCy; the remainder of the Cy being lost, so far as practice goes. It may

form paracyanogen or ferrocyanide, but even then the loss is less than occurs in the zinc-boxes, because of the weaker solutions used. The Pelatan-Clerici process takes advantage of the cyanide liberated at the anode in this way: if free gold is near, the free cyanogen unites with it; or if KCy and Au are in juxtaposition they are also united by the oxygen at that point, and in both instances being electrolyzed, are made ready for deposition. Thus by this process the solving and deposition goes on in the same vessel, in an almost constant electrolyte, until the gold is exhausted.

With the Siemens-Halske process artificial diffusion is necessary to precipitation; but in the above process it is already provided as requisite to the solution; nor is it necessary to enlarge the size of the plates in the latter process beyond a definite proportion necessary for the size of each vat. Hence no inconvenience or extra expense is encountered (p. 758), such as would, in Professor Christy's language, "rob the method of its cheapness, simplicity and convenience."

The high specific resistance, when scientifically utilized, aids in simplifying matters rather than robbing the method of its cheapness.

The haloid salts of potassium are separated into acid and base, the electromotive force necessary to produce decomposition being much less than that required to set free the alkali-metal (*Jour. Chem. Soc.*, 1882, vol. xlii., pp. 260, 353). This being the case, we cannot well accept the theory that $AuCy_2$ goes to the anode and K to the cathode, or that gold under ordinary circumstances is deposited upon an anode of iron from an $AuKCy_2$ solution. It is rarely the case that gold is ever deposited from a cyanide solution on the anode. It is firmly deposited on the cathode.

As to the existence of the auro-cyanhydric acid, and its assimilation to the ferro-cyanhydric acids, the discussion might be prolonged; but an essential character of the ferro-cyanides is that by adding to them a salt of another metal the iron is not displaced and no cyanide of iron is separated.

This is not the case for double cyanide of gold and potassium and for auro-cyanhydric acid, because: (1) chloride of mercury separates the cyanide of gold (see pages 750 and 751 of Prof. Christy's paper); (2) the silver of the silver nitrate precipitates

the gold entirely, making an interchange of metals instead of forming auro-cyanide of silver; and (3) mercury and lead act partially in a similar manner (*ibid.*, page 748, paragraph 4). It matters little about auro-cyanhydric acid, since, if it cuts any figure at all in electrical precipitation, it can be got rid of by using a sufficiently strong current. I am aware that electrical precipitation is not perfect; nevertheless, I think the present practical choice involves, in any case, electrolysis. If we take the zinc-precipitation of the MacArthur-Forrest process, we have voltaic electrolysis, involving one liquid and one metal, the metal depositing upon itself. This means a fouling of the solution and the resulting bullion. If we take the Kendall process of zinc-amalgam precipitation, we obtain a much purer bullion, but Mr. Kendall himself says it is precipitation by electrolysis, in which case we have the voltaic action of two metals, virtually one, in one liquid. The next alternative is the Siemens-Halske method. This is avowedly electrical, involving one liquid and two plates. It gives a good bullion, and will precipitate gold from a very weak solution of auro-potassic cyanide; but it is objectionable by reason of the large amount of Prussian blue formed and the number of lead plates required.

Finally, the Pelatan-Clerici process is electrical, using one liquid and two metal electrodes. The objection to it is that it keeps in use too much mercury, although, on the other hand, practically no mercury is lost.

ALFRED JAMES, Glasgow, Scotland (communication to the Secretary): Prof. Christy lays all cyanide-workers under a heavy obligation by the publication of his exceedingly able paper. This record of results of careful and painstaking work and conclusions deduced from theoretical considerations are of the utmost value to the practical operator, and inspire the hope that Prof. Christy will also publish a similar paper on the mechanical side of the question.

His present paper differs from the majority of previous publications on the cyanide process in that the work therein referred to was based on considerations of theory rather than of practice. One therefore naturally expects that results so obtained will not invariably agree with deductions from practice. Thus, the experimental use of chemically pure cyanide of potas-

sium would lead workers in this direction, as it originally led MacArthur, to design cumbrous though ingenious plants for the admission of oxygen to the working-solutions, when, as Prof. Christy himself shows, for ordinary ores the solutions carry more than enough oxygen for all practical purposes.

It is assumed that in writing of the effect of oxidizing agents the author is referring to experiments with metallic gold and not with auriferous ores, as otherwise the weight of experience appears to be entirely against his conclusions that "each of these . . . . when added to a potassium-cyanide solution increased the solubility of the gold over that shown in a similar solution without such addition." The use of oxidizers has not been found beneficial in practice, except in special cases, such as those in which soluble sulphides are present; in which case, as suggested by Ellis, the oxidizers decompose the sulphides, the retarding action of which he first noticed in 1889. Even in this case it was necessary to select carefully the oxidizer employed.

Perhaps the most brilliant portion of the paper is that relating to cyanogen bromide, in which the author shows that even theoretically oxygen is not necessary for the solution of gold by potassium cyanide, and affirms the important truth that nascent cyanogen is the real factor.

The first generally known record of the proposed application of cyanogen bromide, chloride or iodide to the extraction of gold is that contained in the patent specification of Dr. Gaze, of New Zealand, who applied for a patent for the use of this reagent in 1892; but the knowledge of its real function as a most convenient means for producing nascent cyanogen is apparently of comparatively recent date; indeed, Prof. Christy seems to have been the earliest recorded discoverer.

Some time since an investigation on the subject showed that BrCy was not at all the unstable compound, when mixed with alkalies or alkaline carbonates, that Watts had declared it to be, and that even considerable quantities of caustic alkali or carbonate failed to decompose the compound, whereas, when it was added to a solution of KCy, decomposition commenced immediately.

It was also found that BrCy made by sublimation and BrCy made by the addition of bromine-water to a solution of potas-

sium cyanide behaved alike. The text-book reaction, $KCy + 2Br = KBr + BrCy$, therefore appears to be correct, though with dilute solutions the reaction takes considerable time for completion.

The gradual but complete decomposition of BrCy in presence of KCy with formation of Cy_2 led to the attempt to substitute gaseous CN in solution, with results which showed a distinct improvement with metallic gold over the extraction by KCy alone; but in every case the *nascent* cyanogen proved itself superior.

A point which the author has apparently overlooked is that the decomposition of BrCy in KCy solution, instead of being immediate, is usually very gradual, extending over some hours; the rate depending, among other things, on the purity of the cyanide employed and the amount of alkali present, as it is found that while alkali *per se* does not decompose the BrCy, it greatly accelerates decomposition when KCy is present. A knowledge of this important fact facilitates the addition of BrCy in small quantities from time to time, as required. On the other hand, the tendency to the formation of paracyanogen, even in very dilute solutions, was very great, and the catalytic action rapidly increased on exposure. This probably accounts for the failure in practice to obtain good results with this reagent.

The section of the paper on the precipitation of gold from cyanide solutions abounds in interesting points; but for the present the writer wishes to confine his remarks to Professor Christy's observations on precipitation by means of metallic zinc. At the outset, exceptions must be taken to some of the objections raised by the author (p. 760) against this process.

Objection (1) is well founded and is admitted.

Objection (2) does not appear to be well founded, either upon Professor Christy's theoretical considerations or upon actual results in practice with *dilute* solutions. In the whole of the paper there is not a single reaction shown in which free cyanide is consumed by zinc, with the exception of the hypothetical reaction given on p. 764, and the whole tenor of the previous results on auropotassic cyanide solutions shows that such free cyanide of potassium is entirely unnecessary to the reaction. In every other case it is shown that the zinc merely replaces

the gold in the auropotassic cyanide, and this could scarcely be termed a "great destruction of potassium cyanide."

Moreover, a series of determinations, taken from various plants in different parts of the world, has shown that with weak solutions, as used in practice, there is no appreciable loss of cyanide in the zinc-boxes. The difference in tests taken from the head and tail of the boxes is seldom perceptible, and any difference, when found, is at least as frequently in favor of the tails as against it. (These tests are, of course, not conclusive, and a rapid correct method of determining actual free cyanide in the presence of double cyanide of zinc and potassium and caustic potash is greatly needed.) In this connection it may be noted that even if the whole of the $KAuCy_2$ present in the extractors were regenerated by the action of the nascent hydrogen and in the presence of caustic potash, the increase in the test would be only about .001 per cent.—so slight is the amount.

On the other hand, the assertion that "the great consumption of zinc" is due to free potassium cyanide is untenable in face of the fact that *more cyanide* would be required for this one reaction than is at present *consumed* for the *whole treatment*—that of the ore included. The average consumption of zinc is 0.3 pound per ton treated, and this alone would therefore account for a consumption of cyanide of 1.2 pound per ton. But tailings are being regularly treated in Africa, with an even greater consumption of zinc than that shown, for an actual consumption of only 0.45 pound of cyanide per ton for the whole process.

Objection (3), as to the removal of the zinc and cyanogen residues, is partly met by the usual practice of running water through the boxes immediately prior to cleaning up, and allowing "slimes" to flow through a sieve of fine mesh. Bullion 950 fine and over has been regularly produced by this method; and when the zinc is not conveniently thus separated, an acid treatment is given.

Objection (4) is met by the results of Professor Christy himself, when he shows that perfect precipitation has been obtained even from solutions of auropotassic cyanide *containing no free cyanide*. He objects to the time taken to effect this precipitation. This might have been lessened by increasing the amount

of zinc-shavings; yet it is only about the same as that taken by the electrolytic process. It is also deemed impracticable because 1200 times the theoretical amount of zinc required was present. A rough calculation shows that in all ordinary works, with easily-treated solutions, each box holds about 50,000 times more zinc than is theoretically necessary for the amount of gold in the box at any one time; but one merely runs the solution through continuously until the zinc has completed its function and is replaced by fresh zinc, there being practically no precipitation-limit, as with charcoal. It would appear, therefore, that with such results before him, Professor Christy goes out of his way to assume the necessity of the presence of free cyanide of potassium.

Recent investigations go to prove that effective precipitation is to a considerable extent a question of time of contact; that solutions from which free KCy is absent require longer contact; and that this may be due to the fact that a zinc-gold couple is more energetic in KCy solutions than in solutions of KHO or $KAuCy_2$. This would also explain the greater difficulty of *starting the precipitation* when using very dilute or KCy-free solutions, a zinc-lead couple being almost inert in dilute KHO solutions; whereas, when once the zinc-gold couple is formed, the action proceeds satisfactorily. In practice this difficulty is seldom encountered except when starting a new plant, and may be easily overcome by adding a little free KCy to the boxes at the outset.

Professor Christy objects to certain equations suggested by the writer, on the ground that they bring one back to the point one wished to avoid, and, secondly, that they "fail to explain the increase of free alkali which everyone recognizes as taking place in the zinc-boxes." To deal with the latter point first: In some zinc-box experiments recently carried out by the writer, the amount of alkali present was carefully determined before and after the contact with zinc, with the following results:

	Before. Per cent.	After. Per cent.
(*a*) Two-day test, samples taken every twelve hours.		
Alkalinity (by litmus), ...	0.13	0.11
" (by phenolpthalein) less the difference from litmus-test, ...	0.01	0.006
(*b*) Two-day test, as above:		
Alkalinity (litmus), ...	0.22	0.21
" (phenol.)	0.016	0.014

	Before. Per cent.	After. Per cent.
(*c*) Three day test:		
Alkalinity (litmus), ...	0.168	0.16
" (phenol.)	0.02	0.019
(*d*) Six-day test:		
Alkalinity (litmus), ...	0.168	0.15
" (phenol.)	0.011	0.01

This solution contained practically no free cyanide of potassium, but double cyanide of zinc and potassium was present to the extent of 0.76 per cent. Professor Christy might give some figures in support of his contention as to increased alkalinity, as the above solution or its results may be abnormal.

As to the equations given by the writer bringing one back to $2KAuCy_2 + Zn = 2Au + ZnK_2Cy_4$, Professor Christy even thus fails to show any consumption of *free potassium cyanide*, which was the point the writer started to avoid; but Professor Christy only arrives at this point by making

$$(c)\ 2KCy + 2HCy + K_2ZnO_2 = ZnK_2Cy_4 + 2KHO$$

into a universal reaction instead of a limited one "to a certain extent," as suggested by the writer, it being obvious that in the presence of free KHO, expressly stated to be present, the following reaction would occur:

$$HCy + KHO = KCy + H_2O$$

with regeneration of the KCy, as stated.

In conclusion one cannot but express the hope that Professor Christy may be induced to favor the Institute with a further account of his valuable research on this subject.

(*Later Communication.*)—Since sending the above remarks to the Secretary, I have discovered, by means of experiments involving the crystallizing out of the product from saturated aqueous solutions, that zincate of potash, in the presence of excess of cyanide of potassium, may form double cyanide of zinc and potassium, as stated by Professor Christy. It does not, however, follow that the cyanogen is thereby rendered ineffective for gold-extraction, as Professor Christy would infer. On the contrary, double cyanide of zinc and potassium, in the presence of caustic potash, has a very decided action on gold; and, in fact, the be-

havior of this substance in other ways is so similar to that of free cyanide that it is practically impossible to estimate either of these substances correctly in the presence of the other. The reaction is apparently similar to that occurring with simple cyanide.

$$K_2ZnCy_4 + 2Au + 2KHO + O = 2KAuCy_2 + K_2ZnO_2 + H_2O.$$

FRANCIS L. BOSQUI, Bodie, Cal. (communication to the Secretary): Ever since the cyanide process was found to be an efficient means of gold-extraction, efforts have been made to supplant the original zinc-precipitation method by something "cheaper," "more thorough" and "less inconvenient," in practice. Grave objections have been urged against this mode of precipitation by nearly all writers on the subject, and very recently by Professor Christy, in his excellent paper on "The Solution and Precipitation of the Cyanide of Gold."

The readers of that paper will be chiefly interested in the author's account of his own method of precipitating gold by means of cuprous salts, and in the procedure which he suggests for the recovery of cyanide from strong solutions. While the discovery of this method is unquestionably a very great advance in the chemistry of the cyanide process, it cannot, in all cases, be assumed to possess the practical advantages over the filiform-zinc and electrolytic methods, which the author claims for it.

Professor Christy's four main objections to zinc-precipitation are all, with one exception, unqualified. He dismisses the first three as having "been sufficiently emphasized by everyone who has described the practical application of the cyanide process." The fourth he qualifies thus: "4. The failure, in certain cases, to precipitate the gold." It may be safely inferred here, I think, that Professor Christy means his first three objections to be taken as universal.

It will be of some importance to those interested in the practical development of the process to know that these objections are not universally tenable; and that notably in cyanide-practice at Bodie, California, a pretty exhaustive experience of two years almost completely refutes them.

At Bodie the cyanide-process has been very successfully operated for two seasons of about eight months each. During

the summer and fall there are five plants in actual operation, whose capacities range from 900 to 4000 tons per month; the total capacity of all the plants in the camp being about 11,500 tons per month. These plants are operated on the same general principle, zinc-precipitation being used in all.

The tailings are derived from new accumulations impounded as they run from the mill-batteries, and from old isolated beds in the neighborhood of the town, which have been spared by spring freshets in the long mining history of the camp.

No attempt has been made to keep a record of cost of treatment in all these plants. But such a record has been kept on a total run of 52,665 tons; and to this I shall refer in attempting to show that we have not been embarrassed to any extent in Bodie by the disadvantages which Professor Christy claims for the zinc-precipitation method. I assume, of course, that all the tailings in Bodie are of about the same general character as those of which a record of treatment has been preserved, and that approximately the same results have been obtained from all.

I shall take up Professor Christy's objections in the order in which they appear in his paper.

"1. *The great consumption of zinc compared with the amount of gold precipitated.*"

The record of zinc used at Bodie in the treatment of 52,665 tons of tailings indicates a consumption of 0.22 pound per ton, at a cost of about 3 cents per ton; assuming the price of sheet-zinc laid down in Bodie to be 9.3 cents per pound, and the cost of cutting it about 4 cents per pound. To be more pertinent, there is a consumption of 1.39 pounds of zinc, at a cost of 18.4 cents, per ounce of gold recovered. Theoretically, this consumption is enormous, for it should require only about one pound of zinc to precipitate six pounds of gold; but economically it is very small, since it means a cost of only 3 cents to the ton of tailings.

"2. *The great destruction of potassium cyanide to no useful purpose.*"

It has been often contended that zinc-precipitation calls for the use of stronger solutions than might otherwise be resorted to. Professor Christy stoutly claims for his, and for the electrolytic method, the one paramount advantage that they admit

of the use of very dilute solutions of potassium cyanide. At Bodie, exhaustive experimental tests were made by Mr. C. W. Merrill to determine the proper strength for a cyanide solution, *i.e.* with respect to its extracting power on Bodie tailings; and the conclusion was reached that a solution containing less than 0.1 per cent. cyanide was not so available as a stronger one. The fact then that solutions containing less than 0.2 per cent. cyanide were found unavailable for the best extraction in Bodie, would nullify the chief advantage claimed for both the newer methods of precipitation, at least so far as this particular camp is concerned.

We know that, theoretically, there is a consumption of cyanide during precipitation, that is, during the flow of solution through the zinc-boxes. Some writers have laid particular stress upon this, as being one great source of cyanide-consumption. I have often tried to determine just what this consumption is, by making comparative titration-tests on a solution flowing into, and a solution flowing from, the same zinc-box. I have made these tests very frequently and carefully, but have never found the slightest difference in quantity between the cyanide present in an ingoing solution, and that present in an outgoing solution, from the same zinc-box; the two samples being taken at the proper interval of time. I am aware that the reliability of the silver-nitrate test, as a delicate method of determining the amount of cyanide present in a given solution, has been called into question. But even assuming that the presence of zinc in the solution does vitiate the test to a certain extent, it is hardly probable that the amount of zinc accumulated during a single flow through a zinc-box would be sufficient to destroy the validity of a comparative test. In fact, the quantity of zinc present in 80 tons of strong solution, determined quantitatively at the end of one of our first season's runs, was so infinitesimal as to preclude any such conclusion.

There is still another reason for supposing a comparative silver-nitrate test on ingoing and outgoing solutions to be reliable, namely, the remarkable coincidence that in each case the well-known precipitate came down at precisely the same point in the two test-tubes. In other words, the density of the precipitate was exactly the same in the two tubes, when a common point was reached on the burette. This would hardly occur

with such invariable precision if the test on outgoing solution were really vitiated by the presence of zinc accumulated during a single flow through a box.

The silver-nitrate test is a very simple and serviceable one, and, so far as I know, has been generally retained in practice. Even if it be, in the long run, somewhat vitiated by the presence of zinc in the solutions, it still indicates what might be called a "dissolving-strength," and is, therefore, for all practical purposes, efficient.

The amount of cyanide present in our strong-solution sumps averages from 0.14 to 0.16 per cent. As there is no perceptible consumption of cyanide in the zinc-boxes, it is safe to assume that this deterioration of strong solution is due to the action of "cyanicides" during leaching, and to its dilution by the original moisture in the tailings and by the final wash-water.

All solutions containing more than 0.1 per cent. of KCy are run directly into a "strong-gold-tank"; those containing less, into a "weak-gold-tank." The excess of this weaker solution over and above what is used in the preliminary leaching is run through a series of "waste zinc-boxes," and then to waste. This waste solution averages in strength about 0.05 per cent. KCy. There are approximately 14 tons of it run to waste in 24 hours. This may look like an enormous consumption of cyanide, but the system, on close examination, will be seen to have one great advantage. It prevents an excessive and unwieldy accumulation of strong solution, while the amount of cyanide in the waste is really small, in comparison with what would be required to raise it all to standard strength. This accumulation of weak solution is unavoidable where there is an abundance of water available for wash-water, and where practically all the cyanide-solution in a vat of tailings is displaced before sluicing.

The consumption of cyanide at Bodie on a total run of 52,665 tons of tailings was 0.41 pound per ton; on the first 41,730 tons it was 0.38 pound per ton. The latter figures represent a treatment of 78 tons per day for 535 days. From these data we might indicate approximately just what the consumption of cyanide is in each stage of the process. On a daily treatment of 78 tons the consumption was 30 pounds. The loss of cyanide in spent solution (assuming that 14 tons per day go to

waste, containing 0.05 per cent. of KCy) is about 14 pounds. This leaves 16 pounds per day still to be accounted for. It was ascertained by tests made in percolators on average samples of tailings treated, that the loss of cyanide in actual leaching is about 0.2 pound per ton, or 15.6 pounds per day. This leaves a balance of 0.4 pound per day, or 0.005 pound per ton, which we may assume as the loss of cyanide during precipitation, a very insignificant amount.

These results are summarized as follows:

	Per day, lbs.	Per ton, lbs.
Consumption of KCy in spent solution, . .	14.0	0.179
Consumption during actual leaching, . .	15.6	0.2
Probable consumption in zinc-boxes, . .	0.4	0.005
Total consumption of KCy, ...	30.0	0.384

Of our success in precipitating gold from weak solutions I shall speak in its proper place.

"3. *The great difficulty of removing zinc and cyanogen residues from the gold, thus causing loss in melting, and the production of an unclean bullion.*"

I will admit that this was a great difficulty with us at first, and one which, for a time, seemed insurmountable. It was necessary to devise some means of separating the zinc-sulphate residues from the gold-slimes, after the regular treatment with sulphuric acid. Various forms of filter were tried. A perforated false bottom of wood, packed with sand, was discarded after several trials, on account of the impermeability of the layer of slimes which formed on top of the sand. A filter of asbestos cloth was resorted to, but was likewise discarded. It not only proved to be an imperfect filter, but was found completely disintegrated after the subsidence of the action of the acid, a result probably due to the mechanical untwisting of its fibers during the reduction of the zinc. We were obliged at this time to use a succession of washes, in order to dilute and gradually eliminate the zinc-sulphate—allowing the slimes to settle completely each time. This method, however, was slow and laborious. A Johnson filter-press was next introduced. It seemed to act fairly well on the slimes before acid treatment, but after the destruction of the zinc the gold precipitate appa rently became too finely divided to admit of filtration. A film

of not more than ⅛-inch in thickness on the canvas disks of the press seemed to offer a very complete obstacle to the passage of the clear liquid. The pressure against the disks would run up to over 100 pounds; the press would then have to be opened and the disks scraped. This method was found too tedious, and was finally abandoned.

After many discouraging failures, the following *modus operandi* was adopted, and has since been followed with perfect success:

The slimes and fine zinc are discharged directly from the zinc-boxes into a redwood vat, 6 feet in diameter and 2 feet deep. This vat is protected on the inside by several coats of paraffine paint, has a slight bottom incline for drainage, and is provided with a 2-inch discharge-valve. Here the slimes are treated with sulphuric acid. After the destruction of the zinc, the zinc-sulphate and the slight excess of acid present are diluted by filling the vat with warm water. Within an hour the bulk of the precipitate will have settled to the bottom. The supernatant liquor, to the amount of about 400 gallons, which still contains a small amount of gold-slimes in suspension, is then siphoned off into a 10-ton settling-vat.

The gold-slimes are treated with a succession of these washes, the supernatant liquor being each time drawn off into the settling-vat until the amount of zinc-sulphate remaining in the slimes is insignificant. The liquor siphoned off into the settling-vat, which contains only a very small quantity of slimes in proportion to the total quantity obtained, is left to settle between clean-ups. The clear liquor is drawn off just before a succeeding clean-up, and at long intervals the precipitate is gathered from the bottom and melted.

The bulk of the slimes is finally discharged from the acid-vat into a filter-box. This box is provided, about a foot from the top, with a perforated partition, which is closely covered with two thicknesses of ordinary mill-blanketing. From the compartment beneath this filter the air is withdrawn by means of a steam ejector, and the water is thus removed from the slimes by suction. At the bottom of the box is a one-inch discharge-valve for drawing off the accumulated clear liquid. By occasionally scraping the filter-blankets the passage of water through them is greatly facilitated. These blankets are re-

moved and washed after each clean-up, and clean ones are substituted. The partially dried slimes from the blankets are then completely dried over a furnace and melted in crucibles. The zinc-residues being thus pretty thoroughly removed from the slimes, the difficulties in melting are reduced to a minimum.

Professor Christy lays some stress upon the losses in melting. There is sure to be some loss from "dusting," especially where there is a high draft in the furnace-flues, and where the pots are carelessly charged with the dried slimes.

For some time a considerable value went into the slag, which had to be shipped to the smelting-works; but after a good deal of experimentation, a very suitable flux has been found, which reduces the slag-value to almost nothing. A dust-chamber has been constructed in connection with the melting-furnace, and an effective damper introduced in the course of the flue. The latter is closed at each charging of the crucible, and "dusting" is thus almost entirely avoided.

Our loss in melting has never been more than barely appreciable; and now, since the introduction of a dust-chamber and damper, is wholly insignificant. The wonderfully close correspondence between our actual bullion-yield and the extraction indicated by careful assays of charged and discharged tailings would in itself weaken the supposition of any considerable loss in melting. To be sure, our bullion is low-grade, but we suffer no inconvenience from this except the small increase in cost of transportation and refining in proportion to the value of the bars.

"4. *The failure, in certain cases, to precipitate the gold.*"

Professor Christy probably alludes here, more particularly, to the difficulty in precipitating gold from a weak cyanide solution. He says that in the case of zinc-shavings it is practically impossible to precipitate the gold from such a solution unless it contains one- or two-tenths per cent. of free cyanide of potassium.

In Bodie practice it has been found perfectly practicable to precipitate gold completely from solutions containing as little as 0.05 per cent. of cyanide. Our weak solutions are run through a series of zinc-boxes, of ten compartments each, before going to waste. Each compartment has a capacity of about 8 pounds of zinc-shavings. The solution, after passing through

about 160 pounds of zinc-shavings, is found to be practically free from gold; repeated assays indicating merely a trace.

At the Victor plant, of 50 tons per day capacity, considerable difficulty was experienced in precipitating gold from weak solutions; but by the addition of a third box, making an aggregate of thirty compartments, the trouble was corrected, and most satisfactory precipitation was obtained.

The following table of assays made at random on solution-samples from various zinc-boxes at different intervals in the season's run indicates a high percentage of precipitation in both strong- and weak-solution boxes. These results were obtained by evaporating 250 c.c. (about 8 assay-tons) of the solution in each case, and assaying the residues.

	Strong or Weak.	Value of the Ingoing Solution per Ton.			Value of the Outgoing Solution per Ton.		
		Au.	Ag.	Total.	Au.	Ag.	Total.
1	Strong (0.14 to 0.16 per cent.)..	$4.24	$.45	$4.69	$.05	$.02	$.07
2	" " " " ..	4.39	.43	4.82	.10	.03	.13
3	" " " " ..	4.03	.44	4.47	Trace.	.....	.....
4	" " " " ..	4.13	.46	4.59	Trace.	.....	.....
5	" " " " ..	3.32	.38	3.70	.10	.06	.16
6	" " " " ..	3.05	.39	3.44	.26	.07	.33
7	Weak (0.04 to 0.07 per cent.)..	1.24	.14	1.38	Trace.	.....	.....
8	" " " " ..	2.90	.45	3.35	.07	.07	.14
*9	" " " " ..	3.82	.57	4.39	.25	.07	.32
*10	" " " " ..	4.13	.64	4.77	.21	.08	.29
*11	" " " " ..	1.14	.16	1.30	Trace.	.....	.....
*12	" " " " ..	.52	.15	.67	Trace.	.....	.....

The method which Professor Christy suggests for recovering cyanide from solutions by means of sulphate of zinc might be very satisfactorily applied to spent solutions containing only a trace of gold. The virtual absence of gold from this waste solution might greatly simplify such a method of gold-recovery.

* We have occasionally been puzzled by the appearance in our zinc-boxes of a yellow, aluminous (?) precipitate, which seems to be thrown down in the gold-tanks as soon as the solution leaves the vats. Whenever this accumulates to any extent it interferes somewhat with precipitation. Assays Nos. 9 and 10 were made on solutions taken from a box containing a considerable amount of this aluminous slime. The slightly lower percentage of precipitation is apparent. The two following assays (Nos. 11 and 12) were made on a low-grade solution flowing through the same box, after a complete elimination of the aluminous material, and indicate a normal precipitation.

I do not wish to be understood as condemning the objections which Professor Christy urges against zinc-precipitation. I am aware that in many cases this method has been found an imperfect and expensive one. I simply wish to call the attention of those interested in the cyanide process to one very pronounced exception to Professor Christy's statements, namely, the conditions which obtain in our Bodie practice.

GEORGE A. PACKARD, Butte, Mont. (communication to the Secretary): I see by the *Engineering and Mining Journal* of Aug. 22, 1896, that a patent has been taken out in England for the use of a solution of hydrogen peroxide to furnish oxygen in dissolving gold by cyanide. Early in 1895, after Maclaurin published the results of his investigations (*Jour. Chem. Soc.*, Feb., 1895), I made a number of laboratory-experiments with cyanide, using different oxidizing agents, among them hydrogen peroxide, but, with the ore I was then working on, found no material advantages resulting from the use of any of them, beyond a slignt decrease in the time required to make the extraction.

Later, with another ore, I made two 60-ton tests. With the first 60 tons I used 25 tons of a 0.15-per-cent. cyanide-solution, containing 8.5 pounds of a "15-volume" hydrogen-peroxide solution. With the second 60 tons I used a cyanide-solution of the same strength without other chemicals. The results obtained at the end of 44 hours were as follows:

	Ore assay.	Tails assay.	Percentage extracted.	Cyanide consumed, pounds.
With peroxide, . .	$4.80	$2.25	53.1	51
"Straight cyanide," .	4.40	2.30	47.7	42.5

The amount of peroxide used was all that was obtainable from local druggists and should furnish about 2.9 ounces of oxygen. The amount of gold present in the 60 tons should require, according to the equation,

$$4KCy + 2Au + O + H_2O = 2KAuCy_2 + 2KOH,$$

0.67 ounce of oxygen and 9.52 ounces of cyanide.

Theoretically, there was an excess of over 2 ounces of oxygen in the peroxide, not considering the oxygen dissolved (as air) in

the 25 tons of cyanide-solution used, besides that held in the porous mass of ore. Figuring, however, from the amount of cyanide actually consumed (51 pounds), the amount of hydrogen peroxide necessary, if it was to furnish all the oxygen (3.13 pounds) required by the equation, would be 146.9 pounds of the ordinary "15-volume" solution.

Apropos of the use of oxidizing agents in cyaniding, I give below the results of three comparative 6-ton tests, using sodium dioxide, "straight cyanide," and cyanogen bromide. This ore was not suited to cyaniding, as it contained a considerable quantity of free gold; but the experiments serve to show very well the difference in results obtained by these methods.

Three tanks were filled simultaneously, a car (800 pounds) of ore being dumped into No. 1, the next car into No. 2, the third into No. 3, the fourth into No. 1, etc., thus securing as nearly as possible the same material in each tank. The cyanogen bromide was made at the mill by adding a dilute solution of bromine to a dilute cyanide solution. The ore assayed 0.33 ounce of gold per ton.

Assays, in Ounces of Gold per Ton.

	After 20 hours.	After 40 hours.	After 60 hours.
No. 1, Using sodium dioxide, . .	0.25	0.20	0.20
No. 2, Using "straight cyanide," . .	0.33	0.31	0.21
No. 3, Using cyanogen bromide, . .	0.32	0.24	0.21

The Microstructure of Steel and the Current Theories of Hardening.

Discussion of the paper of Mr. Albert Sauveur. (See vol. xxvi., page 863.)

(Chicago Meeting, February, 1897.)

PROF. A. LEDEBUR, Freiberg, Saxony:* Mr. Sauveur has presented and enriched with original observations a valuable summary of the theories advanced hitherto concerning the hardening of steel; but in one particular, it seems to me, his statements call for correction. He says:

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* Translated by the Secretary.

"The carbon-theory is somewhat crude and incomplete, attributing, in a vague manner, the hardening of steel to the existence at a high temperature of carbon in a hard state and to its retention by sudden cooling. . . . . What this hardening state of the carbon really was, has never been satisfactorily shown. . . . . Is it not difficult to understand how, say, 0.50 per cent. of carbon, be it as hard as it may, distributed throughout 99.50 per cent. of a matrix of soft iron, can produce the glass-like hardness of steel of this composition when suddenly cooled above the critical range?"

I have been always an advocate of the carbon-theory, and have read all the important publications on the subject, and I do not believe that any scientific metallurgist has stated that theory in the way in which it is here presented. A fairer statement would be the following:

In liquid steel, and in steel heated above the critical temperature, the carbon is dissolved like the salt in liquid or frozen sea-water, or like tin in the copper of bronze. If the steel be rapidly cooled below that temperature, the greater part of the carbon remains simply dissolved, and it is then called hardening carbon, because it increases the hardness of the steel the more, the larger the quantity of such carbon present. Yet no one would assert that it is present in a peculiar condition—a "hard state." Mr. Sauveur calls this iron, which contains carbon in simple solution, martensite.

But if the steel be cooled slowly, there results a decomposition (*Zerfallen*) of the iron-carbon solution (alloy), which is the more complete the slower the process of cooling. Accompanied with the liberation of heat, there is formed the carbide Fe_3C, called cementite by Mr. Sauveur, which traverses in a network the general mass. The latter becomes poorer in carbon, and hence softer, the more of its carbon has been consumed in the formation of the carbide. The carbide is hard, and, therefore, the hardness, even of annealed steel, increases with its total contents of carbon; but since the carbide, as a separate body, is distributed through the general mass, it cannot increase the hardness of a whole piece of steel in the same degree as if its carbon had remained uniformly dissolved in the mass.

This theory, which rests upon the investigations of Rinman, Åkerman, Müller, Abel, Osmond and others, I have developed more in detail in various publications, and in my *Handbuch der Eisenhüttenkunde*, and have sought to establish it by chemical

analyses. I still hold it to be the one best fitted to explain the process of hardening, but I do not deny that, besides that most important process, the formation of the carbide under slow cooling, and its decomposition under heating above 700° C., other and subordinate processes are possible, to which may be ascribed the various formations really or apparently observed by microscopists. The microscopic study of the metals is highly fascinating, but it may easily mislead the student into unfounded theories unless he is able to segregate and examine separately the bodies which he believes himself to have detected. For this reason I am not yet able to accord to that method of investigation as much weight as do many of my colleagues, although I am far from denying that its further development is highly desirable, and that every step in that direction should be cordially recognized.

R. A. HADFIELD, Sheffield, Eng.: I regret that pressure of ordinary business prevents my being able to offer a more lengthy contribution to the discussion on Mr. Sauveur's very able paper. If some of those supporting the allotropic theory would follow more upon the excellent lines of Mr. Sauveur, there would be much time saved to both sides in this controversy. He has approached the further consideration of this important subject with an exceedingly fair and open mind, notwithstanding the somewhat strong views he expressed a short time ago in a joint paper. I may be pardoned for saying here that, as it appears to me, joint papers are often not a very wise method of bringing forward conclusions in matters where there must often necessarily be wide divergences of opinion between two investigators, owing to the different lines upon which they work. For a record of work done, a joint paper may be very useful, but when theories are concerned it seems to me unwise for one investigator to attempt to view things in the same light as his partner. It is, therefore, with special satisfaction that I find Mr. Sauveur taking up his own position unbiassed and independent of any other investigator or investigation. I cannot but think that he has done wisely in emphasizing the one *fact* which, to me, seems to subvert entirely the allotropic theory. As it happens to relate to a material upon which I have been working for a number of years, I can entirely confirm his statement, because it agrees

with the facts observed, and because it seems a fair ground upon which to oppose a theory like that put forward by the allotropists, which is entirely speculative, and in support of which no actual facts have been presented. The work done by allotropists entitles them to the highest respect; and that it will be, in fact has been already, of the greatest service to metallurgists, is not only willingly but gratefully granted. But, all the same, not a single fact has been adduced as proving that theory, whereas, on the other hand, the chemical theory, that is, the theory of definite combinations of carbides of iron, while necessarily obscure by reason of the complex questions involved and the still imperfect state of chemical research in this matter, is supported by facts of the greatest importance which cannot be set aside. Workers like Ledebur, Stead, Benneville, Spencer, Hogg, Arnold and many others, who have been studying this complex question for a considerable number of years, and who, owing to the facts discovered, notwithstanding the theories of the allotropists, have ever kept on the even tenor of their way, cannot be set aside.

Therefore, when an alloy like manganese-steel appears to confirm in every way by its behavior both the facts and the theories of those supporting the chemical explanation of the hardening of steel, it should prove a doubly strong confirmation. Manganese-steel, according to the allotropic theories, is hard and non-magnetic, owing to the presence of *Beta* iron; whereas it is quite possible, without diminishing the hardness, to give to manganese-steel a considerable magnetic quality. Therefore, manganese-steel is clearly not a proof of the existence of *Beta* iron. Again, hardened carbon-steel is exceedingly magnetic; and yet, by the theory, the hard iron present should be in the *Beta* form, and, therefore, non-magnetic, which is not the case. Both these cases conflict so entirely with the allotropic theory that, to the writer, it seems useless to consider the latter until its supporters show that these facts can be explained in accordance with their claims.

There is also another iron-alloy which contradicts the allotropists' statement, namely, an iron-nickel alloy containing about 25 per cent. of nickel. Many scientific investigators have very carefully examined this product, especially Dr. J. Hopkinson, F.R.S., of London. He has read numerous papers before the

Royal Society, the Electrical Society and others showing the peculiar reversible properties of this material as regards magnetic qualities. In other words, according to the heat-treatment it receives, it is magnetic or non-magnetic at will. Now, when it is in the non-magnetic condition, according to the allotropic theory, the iron should exist in the hard or *Beta* form; but, as a matter of fact, the alloy is equally soft, and can be readily machined in either condition. It is, therefore, very clear that the allotropic theory entirely fails to explain the peculiarities noticed, and, therefore, must fall to the ground as offering a solution of this important subject.

For my own part, I support the "carbonist" theory, not as an entire solution of our difficulty, but, first, as showing most clearly that the allotropic theory is incorrect, and, secondly, that, whilst necessarily imperfect, it finds such support in the results obtained by the reliable workers I have named, to whom must now be added Mr. Sauveur, as to leave no doubt that it is in this direction we must look for a final solution.

As a "carbonist" I would like to qualify the remarks of Mr. Sauveur (page 902) that we "claim that it is carbon in a hardening state" which offers our explanation. That it is carbon in some particular form or combination I willingly admit; but the exact nature of the combination is still not absolutely defined, though workers like those above quoted, are bringing the explanation nearer every day, whilst allotropists have not a single further theory to advance beyond those put forward several years ago. As regards the facts against them, each year shows some new and more fatal objection to their position.

I trust that one outcome of this year's discussion on this matter will be that the allotropists being asked to explain, or admit that they have no explanation, why, in the two alloys above mentioned, namely, manganese-steel and 25-per-cent. nickel-steel, in the first instance, manganese-steel may become magnetic without losing its hardness, and, in the second case, nickel-steel, a soft material, can become non-magnetic without losing its softness? Both these results are entirely opposed to the allotropic theory. I welcome Mr. Sauveur's paper as being by far the most unbiassed treatise on the subject, and as summing up the position in a way which should bring this controversy to a point where facts, not mere theories, are required.

RALPH G. SCOTT, Monk Bridge Iron Works, Leeds, Eng.: I should like some further light on the statement made on page 888 of Mr. Sauveur's able paper:

Résumé.—Each critical point is accompanied by a structural change which begins and ends with it. In the ranges of temperature where there is no critical point we find no change in the microstructural composition.

Now, comparing Fig. 1, page 866, with Fig. 5, page 887, we find in the first for a steel containing 0.21 per cent. C that Ar_3 begins at 800° and ends at 750°, and, according to the statement just quoted, we should expect to find a change in the microstructure during the cooling of such a steel from a temperature of 800° to 750°; but in Fig. 5 there is no such change indicated. There remains 100 per cent. of martensite through the whole range of 50°.

The next critical point, Ar_2, for steel of the same carbon-contents is shown as beginning at 740° and ending at 700°. Now, in Fig. 5 the change begins somewhere near the upper temperature; but at 700°, where it should (according to the *résumé*) stop, it is in its most active stage and does not get a check until a temperature of 650° is reached.

Referring again to Fig. 1, we have the lowest critical point, Ar, represented as beginning at 690° and ending about 640°, so that the whole of the change from martensite to pearlyte should take place between 690° and 640°. Instead of this, we find in Fig. 5 that from 650° to 620° no change takes place, and that the martensite is not all converted into pearlyte till about 575°, or 65° below the range given in Fig. 1.

I draw Mr. Sauveur's attention to this disagreement in the two diagrams, so that if Fig. 1 is inaccurate it may be rectified, or if there is any peculiarity in the steel represented by Fig. 5 he may kindly explain what it is.

I do not think there is anything from a chemist's point of view opposed to the suggestion of the author that the action of dilute acids upon Fe_3C when it is diffused and in a very fine state of subdivision may be very much more active than the action of the same acids upon plates or grains of the same substance as it exists in pearlyte; but I very much question whether a simple difference of arrangement of the particles of Fe_3C, such as that diagrammatically illustrated on page 905, can at all

adequately account for the difference in hardness between a piece of tool-steel quenched above Ar_3 and the same piece annealed.

Both the quenched and the annealed steel would seem, according to Plate I., Figs. 13 to 16, to contain about the same amount of structurally free cementite.

The question of the absolute hardness of Fe_3C is not gone into in the paper, and it seems to me a very important one. Cementite is spoken of as "intensely hard," "extremely hard," and "the hardest of the constituents of steel;" but no hint is given as to where it is to be placed in the ordinary mineralogical scale of hardness. Is it as hard as corundum? Or does it approach the hardness of the diamond? Still more important to my mind is the question as to *whether it varies* in hardness. If Fe_3C itself varies in hardness, it is useless to explain the hardening of steel by a rearrangement of the Fe_3C it contains, unless we can also explain the cause of the variation in hardness of Fe_3C.

If Fe_3C is always of the same hardness, how is it that a steel containing 23 per cent. Fe_3C (see Plate I., Fig. 16) besides the proportion contained in the pearlyte, can be cut easily by a steel such as that represented in Plate I., Fig. 11, containing only the proportion of Fe_3C diffused, as supposed, through the martensite?

There has been much controversy over the existence of two or three allotropic conditions of iron; and apparently the most apt illustration of allotropy that Prof. Roberts-Austen could give us last year at the Iron and Steel Institute meeting was the case of sulphur, a substance widely different from iron in every way; yet I have not seen any suggestion put forward that the hardening of steel may be due to an allotropic change in the carbon it contains. There is no doubt about the existence of several different allotropic conditions of carbon. One of these is the hardest substance known to us, namely, the diamond.

Prof. Roberts-Austen has carbonized pure electro-iron by contact with the diamond *in vacuo* at a temperature much below the melting-point of iron,* which he considers a clear case of solid diamond carbon being diffused in solid iron without the intermediary agency of any gas.

* *Jour. Iron and Steel Inst.*, 1896, part i., p. 110.

M. Moissan has lately told us* that he has obtained actual visible diamonds by suddenly cooling cast-iron saturated with carbon in molten lead or some other good conductor.

It does not, therefore, seem to me unreasonable to suppose that by treating heated steel in a similar way we may be obtaining a similar result, either preserving some of the carbon in the allotropic condition in which it exists at the quenching-temperature or causing some of it to assume that condition during its quick passage through one or more of the critical ranges.

It might be objected that, on dissolving such steel in acid, we should get a residue of diamond-dust, such as M. Moissan obtained from his cast-iron, which would be recognized by its hardness and other well-known qualities, but the same reasoning applies here as that used in regard to Fe_3C. When nascent hydrogen comes in contact with carbon (no matter what its allotropic condition) so finely divided that it may almost be said to be in a nascent condition itself, it is quite conceivable that hydrocarbons are formed direct at ordinary temperatures. If the idea of a part of the carbon assuming the adamantine form during the quenching of steel is at all tenable, I should then look upon a cutting-edge of hardened tool-steel as formed very much on the principle of the diamond-drill, which consists of diamonds set in a matrix of softer material, the diamonds doing the cutting work, and the efficiency of the tool depending upon the number and quality of the diamonds and the firmness of their setting.

Prof. Henry C. Jenkins, South Kensington, Eng.: In previous discussions upon this subject, the main point of the debate has usually been, whether the allotropic change of state of the iron has anything whatever to do with the peculiar phenomena seen in connection with the hardening and tempering of steel. Indeed, even the occurrence of such a change of state has been disputed. But the change is now generally admitted, and this change coincides so very closely with the temperature at which hardening occurs, and is so modified by it, that it is but reasonable to view the two phenomena as cause and effect unless good ground exists to the contrary. This is the position

* *Comptes rendus de l'Académie des Sciences*, vol. cxviii., p. 320.

of the allotropists who have taken so far, and still take, the position that explorers in a fresh field would occupy. They do not insist, for the present, upon any particular chemical mechanism as that by which hardening is brought about, but only on the view that it is due to allotropic change of state of the iron in the presence of carbon. It may well be that carbides of iron enter into the action of hardening and tempering; it has long been known that Fe_3C occurs in annealed steel, but what is claimed is that one of the properties of the iron, namely, its allotropic change of state, is of as much importance to the phenomena as is the carbon itself.

Much of Mr. Sauveur's objection to the allotropic theory must disappear when it is considered that the critical "points" are really zones that cover even wider ranges of temperature than can easily be followed on the cooling curves; our present means of measuring the progress of the respective changes being imperfect, while the valuable evidence of the microscope itself is only available after the metal has cooled.

Mr. Sauveur has summed up in a most satisfactory way the present state of our knowledge of the facts of the case, and he must be complimented on the very plain way in which he has set these forth in diagrams and tables. But one can hardly follow the details of his suggestion that a diffused carbide is the cause of hardening; for if it be as finely diffused as he indicates, so as to be specially active in a chemical sense when attacked by acids, it must surely be a dissociated carbide, and in no way to be distinguished from the carbon that would be recognized both by the allotropic and by the members of what he terms the "old carbide" school.

Although he would (page 902) forget all the theories, yet he may fairly be asked how the structural changes that set up thermal disturbance in the solid mass can be any other than allotropic ones. But what is in a name? The author appears to be so nearly an allotropist in every other way that the differences can scarcely be seen.

PROF. J. O. ARNOLD, Sheffield, Eng.: I have read Mr. Sauveur's paper with much interest, and cordially congratulate him upon the production of a memoir of great utility, not only in itself, but in the discussion to which it will give rise. Such

a paper could only be produced by an author having a thorough practical and theoretical acquaintance with the highly complex problem under consideration. Although I shall have to strongly oppose some of Mr. Sauveur's deductions (as to facts we seem practically to agree), nevertheless the tone of the paper is such that it is a pleasure to engage in controversy with an author whose object is so palpably the attainment of scientific truth.

Mr. Sauveur says that etching with very dilute nitric acid is unreliable, and has many other objections. As a matter of fact, the beautiful original series of Dr. Sorby and the large collection of micro-sections made during the last seven years at the Sheffield Technical School have all been prepared with very dilute nitric acid. I would suggest that the various methods of preparation may all be made to give good results, and that failure with any one of them may arise, not from the method, but from the manner in which it is applied. For instance, I have not found the iodine-method used by M. Osmond so satisfactory as my own; but it would not be just to declare on that account that M. Osmond's process is unreliable.

Mr. Sauveur reproaches me for not "gracefully" adopting the nomenclature proposed by Mr. Howe and M. Osmond to describe the various micro-constituents of steel. If any doubt existed as to the wisdom of caution in this matter, Mr. Sauveur's paper has finally removed such doubt. There is, I think, no difference of opinion as to the soundness of the general idea that the constituents of steel should be distinguished by mineralogical names, because it is now generally admitted that steel must be regarded as an artificial igneous rock.

Mr. Howe's names: "ferrite" (for Dr. Sorby's "iron") and "cementite" (for Dr. Sorby's "intensely hard constituent") seem admirable. The contraction of Dr. Sorby's "pearly constituent" to "pearlyte" seems also quite free from objection, when an agreement has been arrived at as to the distinguishing features of the constituent thus designated. It will not be necessary to produce before an association of applied scientists like the American Institute of Mining Engineers any argument to prove the fact that steel microscopists can claim the attention of such a society to their work only on the ground of its practical utility in throwing light on the puzzling eccen-

tricities observed in the mechanical properties of steels of like chemical composition. It, therefore, follows that a micro-constituent should be distinguished so as to convey to engineers an idea of the mechanical properties associated with that constituent.

Now, Dr. Sorby's name, "pearly constituent," referred to areas containing distinct alternate laminæ of iron and the intensely hard constituent, Fe_3C. Such areas produce a play of color like mother-of-pearl in a manner analogous to the interference-gratings of the physicists. If nearly pure iron containing about 0.9 per cent. of carbon be microscopically examined under high powers in its ordinary condition, it will be found to consist of iron, throughout which are suspended small, irregular, ill-defined plates and granules of Fe_3C, which present no interference-colors. If the Fe_3C be galvanically isolated, it yields a gray powder. If a bar of such steel be tested in tension, its maximum stress will be about 53 tons per square inch. Under compression-stress it will compress about 30 per cent. under a load of 100 tons per square inch.

If, however, the same steel be thoroughly annealed, the maximum stress in tension falls from 53 to 37 tons per square inch, and at 100 tons its compression is 41 instead of 30 per cent. The microstructure of the annealed bar will consist of beautifully defined, alternate plates of iron and Fe_3C; the latter in different crystals, making well-defined angles, one group with the other. The prepared sample looks almost like a piece of mother-of-pearl. Upon galvanic isolation, the plates of Fe_3C are obtained in bright spangles. Surely two constituents presenting different characters in every respect, except that of ultimate chemical composition, deserve to be specifically distinguished. The names, lamellar and granular pearlyte, do not seem to meet the case, because pearlyte must be laminated to be "pearlyte" at all, otherwise the very finely-divided Fe_3C obtained from steels tempered below a red heat would be also "pearlyte," which is absurd. The wise, and, indeed, *the only just*, course is to call the dark granular areas of normal steels "sorbite." Then the carbide in fully-tempered steels becomes finely-divided sorbite, whilst clearly-laminated areas become fine or coarse "pearlyte." Mr. Sauveur's objection to distinguishing between the carbonized areas of normal and annealed

steels becomes quite incomprehensible when the suggestion is made at the end of his paper that, after all, "sub-carbide" or "martensite" is only iron containing finely-divided Fe_3C. To be consistent, why should this not also be "pearlyte?"

Coming to the name "martensite," it is somewhat difficult to admit the force of Mr. Sauveur's appeal to priority when he has himself abandoned Mr. Howe's name, "hardenite." It is evident that Mr. Sauveur appeals to priority only when convenient, and on other occasions wisely bows to the force of euphony and embraces the sound principle embodied in the words, "*et auctorum plurimorum.*"

As to the suitability of the name "martensite," considered simply as a name, every steel-microscopist will endorse the graceful act of M. Osmond in thus commemorating the patient researches of a distinguished metallographist. But when, as in the case of "pearlyte," an endeavor is made to find out what "martensite" really means, the practical microscopical student turns away in despair.

A mineral must have a typical formula in which, however, the predominant base may be capable of partial replacement by other bases. As a rule, it has also one or more characteristic crystalline forms of approximately specific hardness. Now Mr. Sauveur declares that if iron, containing from about 0.2 to 0.9 per cent. of carbon, be quenched from, say, 1000° C., the whole mass of metal then consists of martensite. But a steel containing 0.2 per cent. of carbon, treated as above, compresses 50 per cent. under a stress of 100 tons per square inch, while a steel containing 0.9 per cent. of carbon, under the same stress, is incapable of compression. The 0.2 steel can be machined as readily as wrought-iron, while the 0.9 steel is almost as hard as diamond.

Mr. Sauveur says also that, when etched, martensite is pale yellow to black in color.

Hence the properties of martensite may be stated as follows: It is very soft and flint-hard; it is pale yellow and dead black; it readily flows under crushing-stress, but is incapable of compression.

What does Mr. Sauveur mean by "diluted martensite?" Such a term destroys the accuracy of his previous definition. The evidence extant points strongly to the following as the ac-

tual facts: Martensite, or sub-carbide of iron, $Fe_{24}C$, can only constitute the whole mass of a steel when the iron, previous to hardening, contained 0.9 per cent. of carbon. This definition applies to pure iron and carbon-steel. When the carbide of iron is replaced, to some extent, by carbide of manganese, tungsten, chromium, nickel, etc., the carbon saturation-point is considerably lowered, so that, in the case of iron containing about 1 per cent. of manganese, the hardened mass consists wholly of martensite (in this case a double lower carbide of iron and manganese) when the carbon present is about 0.7 per cent.

In unsaturated or low-carbon steel, at about 750° C., the martensite and ferrite begin to diffuse one into the other, forming either a solution of martensite in iron or an intimate micro-mechanical mixture of the two constituents.

The constituents, sorbite, troostite and austenite are not within the range of practical discussion, since no one but their sponsor seems to have been able to identify them.

Mr. Sauveur does not see upon what ground I concede to the sub-carbide the right to segregate into distinct areas, and deny the same power to dissolved carbon. My reasons are as follows, and seem to me conclusive: Assume that a steel containing 0.3 per cent. of carbon is treated so that the dissolved carbide is evenly diffused throughout the mass. The steel is then quenched just above the point Ar_1. The areas of martensite and ferrite are distinct—a fact most clearly shown by Osmond's researches. If the carbon were really dissolved, as assumed, then it must be concluded that the dissolved molecules of carbon throw off two-thirds of the molecules of iron in which they were dissolved, and concentrate themselves in separate areas into the remaining one-third of the iron. A segregation of iron and free, undissolved carbon would be a reasonable supposition; yet, so far as I know, graphite or temper-carbon is unknown in unsaturated steels. But, on dissolving the hardened mass in hydrochloric acid, practically the whole of the carbon is evolved as hydrocarbon. Why so, if the carbon was free and in mere solution?

On the other hand, if it be assumed that, under the influence of a higher temperature, the martensite and ferrite diffuse *inter se*, and that, as the temperature falls, *like molecules* segregate to a series of centers, then the thermal phenomena are reasonably

explained, and the evolution of hydrocarbons by the acid is in accordance with well-known chemical laws. The statements made by Mr. Sauveur, that my determination showing that the maximum heat of recalescence is exhibited at 0.9 per cent. C., "is a natural sequence of the structure of the steel"—"has nothing whatever to do with Prof. Arnold's theory"—and "does not point, even remotely, to the existence of his sub-carbide," are certainly sweeping, but are, after all, merely opinions, unsupported by any evidence. What does Mr. Sauveur mean by the *heat of the structural change* at Ar_1? Every one except, as appears, Mr. Sauveur, is satisfied that the heat evolved at Ar_1 is *chemical.* However, Mr. Sauveur's error can be readily proved by practical experiment. The structural change in a carbon-manganese steel, containing, say, 0.25 per cent. of carbon and 1 per cent. of manganese, does not visibly differ from that noted in iron containing only 0.25 per cent. of carbon.

But the heat evolved at Ar_1 from the metal containing manganese is much greater than that observed in the carbon-steel, simply because a new double carbide is formed between the elements, possessing stronger chemical affinities. In the same way a steel containing 2.5 per cent. of chromium and 0.5 of carbon will cause a pyrometric *rise* of 25° C. The thermal portion of Mr. Sauveur's paper is distinctly weak. His last few pages seem to have been written in a hurry. It is alleged that I have written the following sentence: "The point Ar_3 marks the formation of a sub-carbide, while the point Ar_1 is due to the combination of the elements to form the normal carbide Fe_3C." Will Mr. Sauveur kindly locate this quotation? In connection with the hopeless confusion into which (pages 897 and 898) Mr. Sauveur has fallen in connection with recalescence-formulas, will he also state his method for distinguishing Ac_3 and Ar_3 in hard steels? Mr. Sauveur concludes this extraordinary paragraph with the following assertion:

"Prof. Arnold has always contended that the point Ar_3 or Ac_3 marks the formation of the sub-carbide."

This idea must have been evolved from the depth of his imagination; my actual enunciation being as follows: "The heat evolved or absorbed at Ar_1 is due to a carbonization of the sub-carbide, $Fe_{24}C$ to Fe_3C, or a reduction of Fe_3C to $Fe_{24}C$ respectively."

Mr. Sauveur makes some remarks on the crystallization of iron, founding some of his statements on the following misquotation (page 899):

"Prof. Arnold says that if steel crystallized at a higher temperature than 750° C. (Ar_2), say at 900° C., it would imply that all irons quenched at a temperature lower than 900° C. would then have their crystals elongated in the direction of the rolling."

My actual words were: "M. Osmond states that iron crystallizes above 900° C. (the author gives the temperature at about 750° C.). Practical iron- and steel-rollers should understand that M. Osmond's statement involves also an assertion that all iron leaving the rolls at a temperature below 900° C. (a full red heat) has the crystals distorted in a direction parallel to the slag-fibers."

As Mr. Sauveur does not give any experimental evidence in support of his various statements on the phenomenon of crystallization, it seems hardly worth while to go into the matter beyond remarking that the idea that iron heated, say to 1200° C., will on cooling, say to 1100° C., consist of large crystals, the size of which is preserved on quenching, is not likely to find many supporters.

Mr. Sauveur's theory that all the phenomena, physical, chemical and microscopical, are due merely to the variations in the state of division of the normal carbide, Fe_3C, may be supported by a few facts; but the bulk of the evidence is strongly against the accuracy of such a theory. For example, if flint-hard, quenched steel be fully tempered at, say 30° C., the Fe_3C is in a very fine state of division; but the steel is quite soft.

If no molecular change has taken place, but only a finer division of the carbide, why is the specific gravity of hardened steel less than that of normal steel?

In conclusion, I would sincerely thank Mr. Sauveur for his paper, and only venture to suggest that it would have been more valuable had the micro-drawings had reference to steels of specified composition. I also wish to call attention again to the great influence of manganese and silicon on the microstructure of steel, and to the fact that planimeter measurements are only very rough guides to the cubic values of micro-constituents.

PROF. ROBERTS-AUSTEN, London, Eng.: Mr. Sauveur's paper

appears to me to be the most closely reasoned contribution hitherto furnished by those who view the allotropic theory of hardening steel with friendly neutrality, and seek further evidence in its favor before accepting it. An elaborate communication relative to Mr. Sauveur's paper has been kindly sent to me by my friend, M. Osmond,* and, as I am entirely in accord with him, it would be useless for me, in this discussion, to cover again the ground he has been over so carefully.

In Mr. Sauveur's admirably impartial review of the various theories of hardening, there is only one sentence which I wish he had worded differently. He says (p. 893):

"The allotropists had, however, to concede, at the outset, the all-important part which carbon takes in the hardening of steel."

Mr. Sauveur's words, "had to concede," suggest that he does not himself quite realize what an important part carbon plays in the allotropic theory. So far as the promulgation of M. Osmond's theory in Great Britain is concerned, the starting-point was afforded by my British Association lecture (1889) on the "Hardening and Tempering of Steel," in the course of which I insisted that "the presence of carbon is essential to the hardening of steel." The reluctance of the traditionally conservative British metallurgical world to accept the allotropic theory serves to show how thoroughly the importance of the question is understood; and if this were not the case I should feel that I had already wasted too much of my own and of my readers' time in discussing theories about which my own convictions were long ago established.

Directly it is recognized that the carburized irons fall into line with alloys, and these, in turn, with saline solutions, many difficulties disappear. The presence of about 0.9 per cent. of carbon marks a saturation-point of iron by carbon, and everything points to its being the *eutectic* alloy of the steel-series, though it is the eutectic of a *solid*, and not of a liquid solution. A main characteristic of a eutectic is, that it cannot have a formula, because its constituent atoms are not arranged in atomic proportions. Iron, with about 1.0 per cent. of carbon, has the maximum strength of the steel-series—a property which is characteristic of a eutectic alloy; and additional evidence is

* M. Osmond's paper is printed as a part of this Discussion, beginning on page 876.

thus afforded against the view, which Mr. Sauveur so severely condemns, that steel contains a compound having the improbable formula $Fe_{24}C$. It is only when the properties of iron alloyed with carbon, nickel and manganese are correlated that the need for the allotropic theory is felt in full force. It is, however, hardly necessary to point out that, as experimental evidence accumulates, the theory derives more and more support, and the recent experiments of Mr. Howe on the increased tenacity produced by quenching almost carbon-free iron present a case in point.

I may, perhaps, appeal to my own recent experiments on the "diffusion of solid metals," as affording evidence of complex molecular change in solids. I am sure that metallurgists will be grateful to Mr. Sauveur for having so clearly stated the various points at issue.

HENRY D. HIBBARD, High Bridge, N. J.: Mr. Sauveur has rendered an admirable service to steel metallurgists by the clear and concise way in which he has laid down the present status of the various theories relating to the hardening of steel.

The earnest endeavors of the eminent experimenters of all schools in the field have agreed so far that the question is now narrowed down to the one of the composition and constitution of the constituent martensite. As Mr. Sauveur says, if we knew its composition, the problem of the hardening of steel would be solved.

The writer took occasion some time since to point out that martensite has a varying composition, which precludes consideration of it as a definite chemical compound. It seems probable that martensite is an amorphous mass, and that it is, therefore, beyond the power of the microscope to resolve it or to throw light upon its nature.

The theory of Langley and Metcalf is not considered in the paper under discussion, though it is claimed to be a working-theory which accounts for all the phenomena met with in the hardening-process, and for the properties of hardened steel. It seems to me, however, to be partly a case of mistaking one of two effects for the cause of the other.

As I understand it, this theory may be briefly stated as follows: All the phenomena met with in hardened steel may be

explained by considering the piece of steel to be in a state of strain, resulting from the disturbance of the crystalline forces by the carbon which is kept in solution by rapid cooling. A chief point in the evidence which supports this theory is that steel is found to be perceptibly hardened by cooling quickly from the temperature of boiling water. Admitting this, it is still well-known that wrought-iron or low-carbon steel cannot be made very hard, as compared with high-carbon steel, by sudden cooling from the temperatures above Ar_3.

In the case of hardened carbon-steel, hardness and brittleness exist together. Now, every hard substance cooled quickly, and, therefore, irregularly, from a high temperature will be in a state of strain, dependent in some degree upon its mass, and every brittle substance will be in danger of rupture from this cause. Note the cases of glass and tap-cinder. A soft metal suddenly cooled from above W cannot have great strains in it, as it will yield and accommodate itself in a measure to the irregular rate of the cooling it undergoes. Does it not appear, then, that hardened steel has strains because it is hard, rather than that it is hard because it has strains?

The way in which hardened steel articles break spontaneously suggests strains due to irregular cooling rather than to other molecular causes. We know that the cold-working of steel sets up strains within it, and hardens it as well, and we may admit that the increased hardness is due to the strains; but this hardness approaches in but slight degree the flint-like hardness of hardened steel.

The strains in hardened steel no doubt increase the hardness, and this increase it may, perhaps, be possible to determine by experiment on something like the following lines: Harden various samples of steel, anneal them for various periods of time at various temperatures *below* redness, and then determine their relative hardness. This would be merely studying the properties of steel used daily by every blacksmith in the land in tempering cutting-tools. Microscopical examinations of the samples should also be made. It is not likely that any chemical change can take place below a red heat, and the results could hardly fail to be very instructive—if we only had a reliable and recognized test for hardness.

As a speculation, the following may suggest a possible expla-

nation of the formation of martensite. It may be admitted that molecules are perfectly hard of themselves, and that the hardness of bodies depends solely on the arrangements of their molecules, as evidenced by the allotropic varieties of carbon. Above $Ar_{\prime\prime}$, as nearly all agree, the Fe_3C is dissociated. The molecules of Fe and C are then in violent vibration. With slow cooling they reunite during the critical periods, evolving heat as the result of the chemical combination, and form the usual constituents found in slowly-cooled steel. With rapid cooling their motion is suddenly arrested; there is no time for arrangement or combination; and the result is that they remain uncombined, and that *there are, therefore, more molecules per unit of volume* than in the case of slowly-cooled steel. Such a crowded condition of the molecules must result in an amorphous mass of extreme hardness, and may account for the increased bulk and decreased specific gravity of hardened steel.

Some light might be thrown on this matter by the determination of the temperature-coefficient of the electrical resistances of hardened and unhardened pieces of the same steel wire. This is suggested because carbon has a pronounced negative coefficient, in which respect it is unique among the elements. If, therefore, it exists as free carbon in hardened steel, such steel may have a lower coefficient (if not a negative one) than unhardened steel, in which carbon is pretty generally admitted to exist in combination as Fe_3C.

P. H. DUDLEY, New York City: Mr. Sauveur's paper is a very valuable contribution, and will long be studied, not only for the information it gives with regard to his personal work, but because it states so concisely the several theories of the hardening of steel. One must, however, study the papers of the different theorists to form a just conception of the value of their work and their conclusions. That diverse views should be put forth by different co-laborers is not strange. The microstructure of steel is not only complex and difficult at best, but almost all observers must work on materials of somewhat different composition and manner of preparation, with different optical instruments; and only in a general way do they seem to find and see the same structure, which leads them to different interpretations. This must be expected until the subject has

been more fully investigated, and the effects of the different elements, of temperature, and work, in imparting and modifying structure and physical properties, are more completely ascertained.

The practice of calling all elements in steel, except iron and carbon, "impurities" is, in my opinion, no longer desirable. A discrimination should be made, at least in favor of some of the elements, which are as essential as carbon to produce steel commercially. A scientific *consensus* is the final resultant of practical work; and this must constantly be borne in mind, if we wish to make progress. The numbers of kinds and grades of steel, now in use, which are hardened, require any theory of hardening which will cover the general subject to be based upon a broad foundation; and industrial results should be examined, to see how far they confirm laboratory-results.

As Messrs. Osmond and Sauveur point out, the microstructure of steel must be differentiated by polishing and etching, which, even for the plainest and largest structure, require a great deal of skill. It is not yet clear that we can thus resolve and determine all of the minute structure of some kinds of hardened steel. The resolution of the microstructure of steel is limited to the number of lines per inch resolvable by the aperture of the objectives employed.

The table given below, comprising items selected from the

Aperture-Table for Objectives.

Objective. Focal Length.	Aperture. In.	Maximum Limit of Resolving-Power in Lines per Inch.		
		White Light.	Blue Light.	Photography.
1 inch	0.25	24,103	26,126	31,749
½ "	0.60	57,846	62,702	76,199
¼ "	0.80	77,128	83,603	101,598
⅜ "	0.90	86,769	94,053	114,298
⅜ "	1.00	96,410	104,503	126,998
1/10 "	1.20	115,692	125,404	152,397
1/10 or 1/12 inch	1.30	125,333	135,854	165,097
1/12 inch	1.40	134,974	146,305	177,797

aperture-table published by the *Journal of the Royal Microscopical Society*, gives the maximum resolving-power in lines per inch for objectives of different aperture, for white light, blue light and photography. It represents only a few objectives, but will

serve to illustrate my proposition. I have placed opposite the apertures the usual or frequent focal length of the objective.

The one-inch objective given in the table has a much larger aperture than usual, and will stand deep eye-piecing, giving sharp definition for a high magnification, which, however, does not increase beyond a certain limit its resolving-power, which depends upon its aperture. It is resolving-power and good definition, as well as magnification, that are required to see delicate structures. I have given in the table the theoretical number of lines for an objective of 1.40 aperture, but there are only a few of these yet in use.

Theoretically, a great deal is to be expected from the recent "apochromatic" objectives, made by Zeiss, of Jena; and doubtless they will give better results than those obtained with other objectives, when greater experience has been acquired in their manufacture and use for special purposes, such, for instance, as examination of steel under vertical illumination. Hitherto, the photomicrographic work of the apochromatic objectives on steel has not, so far as I know, equalled that of some of the older objectives. I understand that the maker says they will need a special correction for such work.

Osmond's high-power work and his unequalled high-power photomicrographs were made, as I understand, with Nachet et Fils No. 9, $\frac{1}{12}$ homogeneous objective, 1.30 aperture. I use a homogeneous objective, one of the best made up to a few years ago, of 1.20 aperture. This, of course, cannot do the work shown by Osmond, for the actual resolving-power often falls below the theoretical; a portion of the structure may not be in the focal plane of best resolution, and is partially lost. The large-aperture homogeneous objectives are only recent productions and excel in performance anything supposed to be possible ten years, or less, ago.

The more one studies Mr. Osmond's papers, describing what he has found in his careful experiments at definite recorded temperatures, the more certain one must feel that he has seen what he describes; and his conclusions carry great weight, even though it be not possible, with different means and material, and without his great skill and experience, to reproduce what he has so clearly and candidly described. When he declares that he finds iron to be allotropic, he must have microscopic evidence for such a declaration. Does the indus-

trial use of iron furnish any corroborative evidence, indicating such allotropy? If we consider, for instance, the failure of iron car-axles, of which there are over 5,000,000 in use on steam railroads alone, the failure of many of them can only be explained, it seems to me, by allotropic conditions of the iron in the same axles. The treatment in service being alike for each end of the axle, carrying the same percentage of load, one end fails while the other does not. Ninety per cent. of the axles from the same lot will run and complete the limit of service expected.

I have taken several axles condemned for further service and subjected them to a drop-test of 1640 pounds falling 20 feet, the supports being 3 feet apart; and while the center would sustain the drop, one of the journals would fly off, the fracture being entirely new. But after taking the axles, heating the journals and wheel-seats to a dull red and cooling them, I could batter them all out of shape before fracture would take place.

Iron can be hardened by quenching, cold-hammering and rolling; though, as I need hardly add, the hardness thus obtained does not equal that which is secured by hardening steel. The heads of iron rails, after a long service in the track under light traffic, have a thin surface-layer which has become very hard—reaching 27 on the scale of the "pyramidal cutter," or about the hardness of steel rails when new.

Mr. Sauveur says: "Martensite is that constituent which exists at a high temperature, and being retained by sudden cooling, confers hardness upon quenched steel. . . . Of its real composition, however, we have no direct evidence." So far as my investigations indicate, this hardness is only a condition which is found in the martensite, rather than a permanent quality of the martensite itself. My conclusion is based on the following reasons: 1. The different grades of hardness which are produced by different contents of carbon; 2. The higher heat to which the lower-carbon steels must be subjected for diffusion of the carbon before hardening; 3. The modifications produced by tempering, without apparently modifying the structure, except to relieve it of internal tensions.

What does take place in the martensite is as yet very obscure.

To ascertain whether any indications of change in structure would be given by increase or decrease of length of bars after

hardening, I took three ½-inch square bars of tool-steel, about 7 inches long, and measured them before and after hardening. The ends of the bars were finished to ¼-inch square, measured, and capped with fire-clay to prevent oxidation during heating. They were heated just above the critical range and hardened. The fire-clay dropped off a few seconds before the heating was finished. There was no evidence that any scale had formed and flaked off during the hardening. Two of the bars had apparently been shortened, and one slightly lengthened.

A bar of Jessop's special tool steel, ⅝-in. square, was taken, and the ends were finished to ¼-in. square, the length being 11.6682 inches. The ends were capped with fire-clay and annealed—the length then measuring 11.6742 inches. The bar was heated to dull red and hardened in water, and then measured 11.6694 inches.

Three bars, 1 of "titanic" tool-steel, ⅝-in. square, 1 bar of 0.60 per cent.-carbon rail-steel ⅝ by ¼ in., and a third of 0.60 per cent.-carbon rail-steel, were taken to a spring-maker who had a muffle for heating. The rail-steel was from test-bars rolled from the small test-ingots. The bars were all finished to ¼-in. square on the ends; the ends were capped with clay before heating in each case; and they were all hardened in oil and tempered to a spring-temper. The pieces were not annealed before hardening but were left as they came from the hammer and rolls. The results of the measurements are given in the following table:

Kind of Steel.	Length in Inches.				
	Original.	Hardened.	Shortened.	Tempered.	Lengthened.
Titanic, ½ inch square	11.4367	11.4350	0.0017	11.4351	0.0001
60-carbon rail, ⅝ by ¾ inch	11.4932	11.4943	0.0019	11.4945	0.0002
60-carbon rail, ½ inch square	11.4269	11.4250	0.0019	11.4251	0.0001

These figures are offered only as approximations obtained from shop-experiments, and as indicating the desirability of more accurate measurements to determine the volume of steel as affected by structural change under heat-treatment. If the steel is heated up much above the critical range, and remains so for only a few minutes, a coarser structure is formed,

decreasing the density of the steel when hardened, as well as its strength.

The heating of steel during manufacture and subsequent forging has always been justly regarded as one of the steps which can not be neglected for fear of over-heating the steel.

In a piece of steel heated to visible redness, slightly below the point of recalescence, the mobility of the carbon is possible; and, just above the point of recalescence, the mobility of the constitution is still sufficient to rearrange the preceding structure, unless a pretty well defined structure has been developed at a higher temperature. The steel is plastic and can be shaped by forging or rolling without injury to the metal. But if the temperature be raised still higher, and if time permits, the crystalline forces build up a constantly increasing size of mineral aggregates; the steel loses some of its plasticity, and will be injured, more or less, at least in the first stages of forging or rolling. Experience does not sustain the conception that steel does not acquire a crystalline structure on a rising temperature.

PROF. E. D. CAMPBELL, Ann Arbor, Mich.: I have studied Mr. Sauveur's paper with great interest, and appreciate the clearness with which he has described the current theories of hardening steel. During the past six years I have myself been attempting to obtain some results which might throw a little light on this question, and have personally come to the conclusion that we must regard steel in the same light as organic chemists regard the bodies which they investigate.

As early as 1864, Hahn* obtained, from different cast-irons, a variety of hydrocarbons, a part being gases, belonging to the olefine series (*i.e.* having the general formula C_nH_{2n}); a part gases of the paraffin series (general formula C_nH_{2n+2}); and a part unidentified liquid hydrocarbons. Later, in 1877, Cloëz† dissolved a large sample of manganiferous white iron, and recovered part of the carbon by absorbing the gaseous olefines in bromine, thus converting them to the di-brom derivatives; this absorption taking place according to the equation: $C_nH_{2n}+Br_2 = C_nH_{2n}Br_2$. In addition to the carbon obtained in the above manner, Cloëz obtained also a portion of his carbon as low-boiling-point liquid paraffins, especially $C_{12}H_{26}$, together with some unidentified liquid hydrocarbons. Both Hahn and Cloëz

* Liebig's *Chem. Ann.*, cxxix., 57. † *Compt. rend.*, lxxxv., 10.3.

proved qualitatively that the olefines,—ethylene (C_2H_4), propylene (C_3H_6), butylene (C_4H_8) and pentylene (C_5H_{10}),—and di-butylene (C_8H_{14}) were present in the gases; and Cloëz identified the paraffins from decane ($C_{10}H_{22}$) to hexa-decane ($C_{16}H_{34}$). Both mention that duo-decane ($C_{12}H_{26}$) was the predominant member of the oily paraffins. The work of neither of these chemists was strictly quantitative; and neither attempted to account for the total amount of carbon, or to deduce from their results any generalizations which might lead to a theory concerning the relation of carbon to iron.

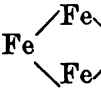
So far as I am aware, not much work has been done in the way of studying the products of solution of steel, by dissolving the metal in hydrochloric acid or sulphuric acid, and attempting to identify the hydrocarbons produced. Yet, when we consider the methods used in organic chemistry for deducing the structural constitution of bodies, it would seem that we can not expect to get at the true constitution of steel until we employ analogous methods. In organic chemistry, the structure of the vast number of organic bodies has been determined by depending almost entirely upon the products obtained by the action of reagents upon each body; and if the chemists in organic work are justified in assuming that the constitution of a body yielding certain products of solution can be deduced from the products themselves, then it would seem to me that metallurgical chemists are not only justified in following the same procedure, but must follow it if they expect to obtain results at all comparable with those obtained in organic chemistry.

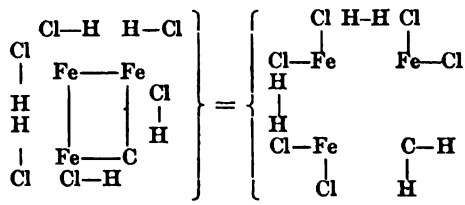
In every theory at present held with regard to the constitution of steel, the formula for the carbide of iron, obtained from annealed steel, is given as Fe_3C. The advocates of all the theories, whether they be carbonists, allotropists, carbo-allotropists or supporters of the sub-carbide theory, assign to the carbide from annealed steel the above formula; and yet no one, so far as I am aware, has attempted to write an equation which should give an idea of the manner in which that carbide dissolves in hydrochloric acid. They have all been satisfied to assign a merely empirical formula to the compound and have made no attempt to study the structural constitution of the body in question. In the course of my own work I have felt, more and more strongly, that the properties of steel do not de-

pend upon the presence of a body of so simple a formula as Fe_3C, but that the relation of iron to carbon is much more complex than would be indicated by such a formula. During the past three years especially, I have been attempting to determine the products of the solution of steel in hydrochloric acid, and to study the influence of the percentage of carbon and of varying heat-treatment upon these products; and although I am not at present prepared to give the details of all the results obtained, since the research is not yet, and will probably not be for some time to come, complete, yet I can state in a general way what I think is indicated.

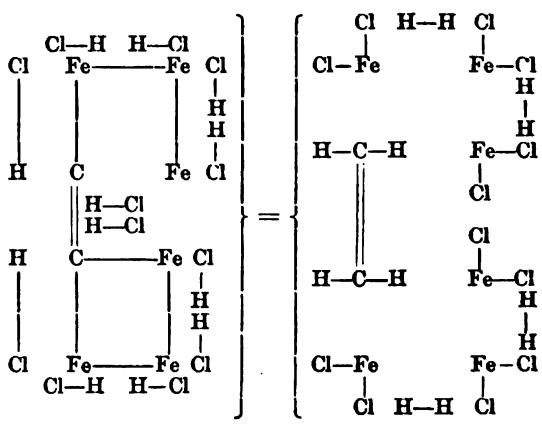
Last year we obtained from a sample of annealed steel (containing 1.28 per cent. C), by treating it electrolytically in dilute hydrochloric acid, about 55 grammes of a practically pure carbide of iron, agreeing almost exactly with the empirical formula, Fe_3C. This carbide gave as its products of solution: hydrogen; probably ethane (C_2H_6); and a large amount of butylene (C_4H_8); and part of the carbon was unaccounted for, but the odor of the solution gave almost positive evidence of some of the lower-boiling liquid paraffins, probably butane (C_4H_{10}) or pentane (C_5H_{12}). A detailed description of this carbide may be found in the *American Chemical Journal* for December, 1896. If we consider the results of the solution of the above carbide and the products obtained by Hahn and Cloëz, the conclusion that the carbide of iron capable of yielding such complex products has a formula of Fe_3C, does not seem tenable. But if we bear in mind two well-known facts, there seems to me to be a hypothesis capable of explaining these products of solution in a manner in perfect accord with well-known chemical laws. The first of the facts to which I would call attention is the well-known tendency of many of the olefines, especially those of higher molecular weight than propylene (C_3H_6), to polymerize or condense. The second fact is that the olefines have a tendency to combine directly with nascent hydrogen, to form the corresponding paraffins, this addition of hydrogen taking place according to the equation $C_nH_{2n}+H_2=C_nH_{2n+2}$. This is especially marked in the case of ethylene (C_2H_4), combining with hydrogen (H_2) to form ethane (C_2H_6).

Bearing in mind the two above facts, I would offer the hypothesis that the carbide of iron, isolated from annealed steel, has

a formula, not Fe_8C, but much more complex. I am inclined to think that iron, like hydrogen, is capable of forming a series of what I think we might justly term ferrocarbons; that these ferrocarbons have a general formula C_nFe_{3n}; that the group  replaces two hydrogen atoms in hydrocarbons of the olefine series. Whether a ferrocarbon of so simple a structure as , the first member of the series, can exist, will probably remain undetermined; for if such a ferrocarbon existed, the first products of solution in hydrochloric acid would be according to the graphic equation:



But this would give methylene, which, up to the present time, has never been isolated, and which, owing to its unsaturated carbon would immediately combine with hydrogen, probably not stopping short of complete saturation to methane (CH_4); or, more probably, it would first polymerize and combine with hydrogen to form ethane (C_2H_6). The next member of the ferrocarbon series (C_2Fe_6) would yield as its primary products of solution the first known member of the olefines, ethylene (C_2H_4), according to the equation:



The ethylene would take up more or less hydrogen to form ethane. In the same way we can very readily understand the structure of the other members of the ferrocarbon series, C_3Fe_9, C_4Fe_{12}, etc., and how these members, by the action of hydrochloric acid, might yield the corresponding olefines, which, in turn, by condensation and partial hydrogenation, would complicate the products of solution, rendering them, perhaps, even more complex than the ferrocarbons from which they were derived.

With regard to the bearing of this hypothesis of a series of ferrocarbons on the theory of the structure of steel, I might offer a few suggestions.

It is generally recognized, I believe, that what Prof. Arnold has properly called the saturation-point of carbon for iron is reached when the carbon is present to the extent of 0.9 per cent., or when the metal has the empirical formula CFe_{21}. I need not enlarge on the fact that the maximum influence of carbon on steel is felt when this point is reached. In annealed steel of this composition, we have the entire mass consisting of pearlyte. When the metal is heated above Ar_1 this is all changed to martensite. If the steel contains more than 0.9 per cent. of carbon, we have free cementite, associated with pearlyte in annealed, or with martensite in hardened, steel. If the carbon be much below 0.9 per cent., and the metal be quenched from a temperature just above Ar_1, Mr. Sauveur, as well as others, has shown that we have martensite, occupying the same space as the pearlyte from which it was derived, and accompanied by ferrite. If, however, the steel containing less than 0.9 per cent. of carbon be heated to higher and higher temperatures before quenching, the martensite occupies, as has been shown so ably by both Sauveur and Osmond, a greater and greater proportion of the mass, but gives different shades of color on treating the polished specimens with iodine or nitric acid. It is principally over the changes which take place on heating above Ar_1 that the differences of opinion among metallurgical chemists have arisen; and it is with a good deal of hesitancy that I would offer a sixth hypothesis in regard to some of these changes, and their influence on the physical properties of steel. First, I would suggest the probability that the carbon in annealed steel occurs as one or more, usually several, members of the ferrocarbon series, C_nFe_{3n}, and that these ferrocarbons crystallize out

of solution at Ar_1, uniting at the same point with Fe_{18n} of iron of crystallization; but that this Fe_{18n} is not ferrite, as is usually assumed. For if this iron of crystallization, which constitutes the alternate laminæ of pearlyte, were ferrite, it should exhibit the same behavior as ferrite towards nitric acid and iodine, under treatment in a polished section; but it does not. We are thus led to regard the constitution of pearlyte as C_nFe_{3n}, Fe_{18n}. What is the value of $_n$ in different steels, is the point on which I have been at work for the past three years; and my results so far go to show pretty conclusively that, in steels with less than 1.3 per cent. of carbon, the dominant ferrocarbon is C_4Fe_{12}; but the derivatives obtained have also shown that C_2Fe_6, C_3Fe_9 and C_5Fe_{15} are almost certainly present, and that, with increasing amounts of carbon from low-carbon steels up, the amount of C_2Fe_6 in annealed steel increases. Further (although on this point we have but one set of results), the indications are that in the given steel in the hardened condition, there is more C_2Fe_6 than in the same steel annealed. Keeping in mind the fact that, qualitatively, the products of solution of hardened steel are identical with those of the same steel annealed, it would seem almost certain that when steel is heated above Ar_1, the ferrocarbons (or, as is usually said, the carbide of iron) do not dissociate but pass into solution in the steel. We might regard the absorption of heat at Ac_1 as being due to the first step, namely, the solution of the ferrocarbons in their iron of crystallization, exactly analogous to the melting of many salts in their water of crystallization, and thus expressing the conversion of pearlyte into martensite, occupying the same space, or, in other words, into Pr of Arnold's sub-carbide.

As to the causes of the other two critical points Ar_2 and Ar_3 there is still room for speculation. Our own study of the products of solution of steel indicate, although we have not carried them far enough to prove them positively, that the ferrocarbons dissociate more or less as the temperature rises. This we might naturally expect, since in low-carbon steels the ferrocarbons diffuse more and more through the free ferrite. Whether Ar_2 and Ar_3 mark points at which certain of the higher members of the ferrocarbons split up, as for instance C_4Fe_{12}, splitting into $2C_2Fe_6$; or whether the two upper critical points are indicative of allotropic changes in the iron, I am as yet unprepared to ex-

press an opinion. But what I do think is, that the ferrocarbons in low-carbon steel are diffused, as would naturally be expected, more and more perfectly with rise of temperature; that this diffusion is probably accompanied by a splitting-up of the higher into the lower members of the ferrocarbon series; and that, when the metal is quenched, these ferrocarbons remain dissolved in the mass of the metal, imparting to it certain properties. One of the strongest arguments to my mind, in favor of an allotropic change of the iron at high temperatures, is the intense hardness, accompanied by brittleness in quenched steel, and the fact that part of this hardness and most of the brittleness disappear on reheating the metal to a temperature of about 250° or 300°C., a temperature below that at which polymerization of the ferrocarbons begins. Moreover, the metal which has its maximum magnetic hardness in the quenched state becomes slightly magnetically softer when reheated to 200°C.; then suddenly very much softer magnetically, when heated to 300°C.; and continues to become slightly softer still as it is reheated, until the recalescent point is passed and the metal assumes its minimum magnetic hardness in the annealed state. Thus we might see a relation between the magnetic hardness of steel and a possible allotropic condition of the iron; since the great loss of magnetic hardness seems coincident with the loss of brittleness between the temperatures at which steel is usually tempered. On the other hand, the retentivity of magnetism, or the persistence with which magnetism, when once acquired, is retained, does not fall off at 300°C. as does the magnetic hardness, but increases steadily, though slightly, up to the point of 600°C., or nearly to the recalescent point. Of course, when the recalescent point is passed, and the dissolved ferrocarbons crystallize out, we have annealed steel, and the retentivity is lower than in any other condition.

These results would indicate that while hardness and brittleness in steel may not improbably be due to an allotropic form of iron, the increase in tenacity, elasticity, retention of magnetism and other properties characteristic of tempered steel may not improbably be due to the presence of dissolved ferrocarbons. The iron in these ferrocarbons may possibly be in an allotropic form; but the assumption of such allotropy would not be necessary for an explanation of the properties observed.

Another argument in favor of the existence of a series of ferrocarbons and of the splitting-up of these bodies when steel is heated, is the behavior of steels with nitric acid in the ordinary color-carbon-tests. The color of the nitro-derivatives obtained by solution of the higher members of the series would in all probability be darker than that obtained from lower members, just as the bromine-derivatives of the higher members are darker than those of the lower. This would show why, in making color-tests for carbon, it is necessary for accuracy that not only the standard and the unknown should have approximately the same percentage of carbon, in order that the members of the ferrocarbons should be the same, but, in addition, that they should have undergone the same heat-treatment, for the same reason.

Since hydrogen may be regarded, in its chemical behavior, as a metal, and since carbon is capable of forming so great a variety of compounds with that element (compounds in which many carbon atoms are linked to each other in almost numberless ways), it would not seem improbable that iron is capable of forming a number of ferrocarbons analogous to the hydrocarbons; and I can see no inherent reason why a number of the other metals, whose carbides have been regarded heretofore as of uncertain composition, may not likewise form series of what might properly be called metallo-carbons.

Our work at present is confined to the ferrocarbons, in an endeavor to establish the hypothesis of their existence by studying the products of solution and the thermo-chemical reactions of iron and steel. If we succeed in establishing the probable existence of ferrocarbons, we hope to take up in time some of the other metallocarbons, and, if possible, get metallurgical chemists to realize the fact that inorganic chemistry must be studied in an entirely different way from the methods at present employed; that the physical properties of metals depend more upon the structural constitution of the components than upon the ultimate composition of the mass.

F. OSMOND, Paris, France: Mr. Sauveur has summarized very clearly and accurately the results obtained in recent years by studying both the critical points of different varieties of steel and their microstructure. As regards his method of

preparing the samples and his examination of them by the microscope, I have no observations to offer, and I am as fully in accord with him as it is possible to be. The divergence between us only begins in the interpretation of the results.

These divergences are, however, useful. I am not one of those who regret the multiplicity of the theories; the discussions to which they give rise, the experiments to which they lead, and the considerations they provoke are conditions of scientific and technical progress.

Examination of Mr. Sauveur's Theory.

Mr. Sauveur considers it to be experimentally demonstrated that each critical point corresponds to a change in structure which begins and ends with it. Starting thence, he asks whether these changes of structure may not even be the cause of the critical points, and he accounts for the physical, chemical and mechanical phenomena by a different distribution of the same carbide of iron, Fe_3C, throughout the mass of iron.

The fundamental proposition is, that "each critical point is accompanied by a structural change which begins and ends with it. In the ranges of temperature where there is no critical point we find no change in the microstructural composition." This appears to me to be perfectly accurate for steels of medium carburization, say between 0.30 and 0.90 per cent. of carbon. These limits may, however, vary with the chemical composition. But I do not find that the proposition is completely sustained in the case of steels which have either very little or very much carbon.

I have studied in detail the variations of structure in a metal containing:

C, 0.14; Si, 0.045; S, 0.018; P, 0.01; and Mn, 0.19 per cent.;

and I find the following figures for the microstructural composition, neglecting the rare transitional forms:

	Per cent. Martensite.	Per cent. Ferrite.
Heated to 960° C. and quenched at 670°, between Ar_1 and Ar_2,	14	86
" " " " 770°, " Ar_2 and Ar_3,	24	76
" " " " 820°, middle of Ar_3,	46	54
Heated and quenched at 1000°,	61	39
" " " 1340°,	90	10

These figures, it must be understood, are approximate; but

they are such as do not leave room for any doubt. The changes of structure are not arrested, during heating, at the point Ac_1; the martensite has continued to spread far beyond it. Time unquestionably intervenes, as is shown by the results of the experiments of Mr. Howe* and of M. Charpy.† It is possible that the diffusion of carbon might be complete when Ac_3 has been reached; but this diffusion is not instantaneous and demands time, which becomes proportionately longer as the temperature is lower. In reality, the formation of martensite during heating is a true internal "cementation" of the segregated ferrite, and it is well known that cementation is not a rapid operation. This is not clearly perceptible in highly carburized steels, because the mean distance between the lamellæ of cementite is not, in round numbers, more than 0.001 mm. But in low-carbon steels, in which the distance between the carburized kernels of pearlyte is about one hundred times as great, it is not surprising that the regular diffusion of the carbon demands a much longer time. However this may be, the fact that the variations of structure do not, in certain cases, end with the critical points, proves that the structural variations are a consequence of the molecular changes which occur at these critical points. It is as if a mixture of ice and a soluble salt were heated; if the fragments of ice and salt are infinitely small and perfectly mixed, the solution will be homogeneous as soon as it is melted. If, on the other hand, the fragments are large and irregularly distributed, this will not be the case, at least not unless the solution is stirred. The diffusion of the salt dissolved in the solvent will proceed according to known laws, starting from points of maximum concentration. In any case the diffusion of salt in water presupposes the preliminary melting of the ice; it is a sequel of this melting, and not its cause. In the absence of salt the melting would occur at a slightly higher temperature; but it would equally occur.

In the highly carburized steels, those which, after cooling, contain segregated cementite, analogous phenomena are recognized.

According to Mr. Sauveur, the proportion of segregated cementite in these varieties of steel remains the same, whatever

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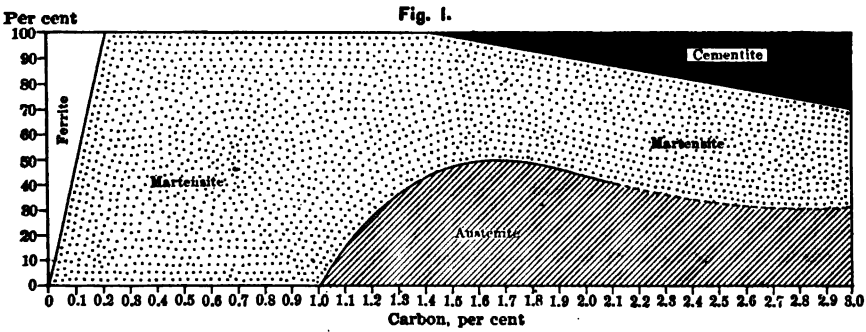
* *Trans.*, xxiii., 508.

† *Comptes rendus*, vol. cxviii., p. 420 (February 19, 1894).

may be the quenching-temperature above $A_{3,2,1}$, and it would seem that it practically is so provided that the quenching is effected under the usual conditions. But these conditions allow time for important changes to be effected. If the cooling is rapidly conducted by quenching small samples of steel in water mixed with ice, the cooled metal should reveal a structure which more nearly approaches that possessed by the metal while it is red hot, and it can be established that the proportion of segregated cementite varies with the temperature of quenching. If this temperature exceeds 1000°, a percentage of carbon amounting to 1.60 may be reached without isolation of cementite. At the same time a new constituent appears,* for which Prof. Roberts-Austen has kindly accepted the position of godfather; and the properties of this constituent, so far as it is possible to

Fig. 1.



Microstructural Composition of Steel Quenched from about 1050° C.

ascertain them in a mixture, closely recall the properties of steel with 12 to 13 per cent. of manganese and steel with 25 per cent. of nickel. Its most salient property is that of being scratched either by a darning-needle or by apatite; and this fact the partisans of the heroic action of carbon have not as yet even attempted to explain. I have not hitherto succeeded in raising the proportion of austenite above some 50 per cent.; when the percentage of carbon exceeds about 1.60, cementite reappears.

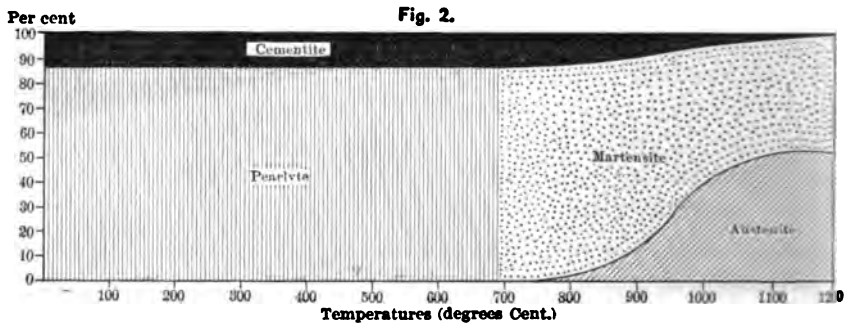
To sum up the matter, while Mr. Sauveur represents the microstructural composition of steels quenched above Ar_3 by the upper diagram of his Fig. 4, I believe that I can represent it, under the special conditions of quenching already defined, by the diagram here given in Fig. 1. This is, however, a some-

* *Comptes rendus*, vol. cxxi., p. 684, November 11, 1895.

what crude approximation, and should be rendered more accurate by numerous experiments. For the same steel, with 1.50 to 1.60 per cent. of carbon, the microstructural composition would be represented in terms of temperature of quenching by a diagram such as Fig. 2. In other words, the proportion of segregated cementite, which should be some 13 per cent. in the steel in question, quenched above $Ar_{3,2,1}$, may vary from about 13 per cent. to almost none. This shows that the cementite continues to be dissociated, or at least to be dissolved progressively, when the temperature is progressively raised above the critical point.

As I see absolutely nothing among the observations of Mr. Sauveur which is in contradiction with my own, I hold the latter

Fig. 2.



Microstructural Composition of Steel Containing 1.5 to 1.6 per cent. of Carbon, and Quenched at Various Temperatures.

valid; and I conclude that the changes of structure, not being necessarily restrained at the critical points, are not the cause of these points.

The opinions of Mr. Sauveur, moreover, from the triple point of view of chemistry, physics and mechanics, raise difficulties which he has been the first to recognize and very frankly to point out.

From the chemical point of view the minute state of division of the carbide Fe_3C explains sufficiently well the readiness with which it could be attacked by hydracids, because plates of dimensions which do admit of measurement appear to be a little attacked. But in the colorimetric estimation of carbon by Eggertz's method, why is it that in hardened steel there is a certain amount of carbon which is apparently lost? The fine

state of division of the carbide may well render the attack of the acid more rapid; but why should the *nature* of the reaction itself be changed?

From the physical point of view, the carbide Fe_3C, which is supposed to be stable at all temperatures, may well be compared above A_1 to a gas, the expansion of which produces cold and its condensation heat, and consequently critical points. But as concentration towards one point of the mass presupposes rarefaction at a neighboring point, the two phenomena, represented by contrary signs, should establish equilibrium or nearly so. In the same way, if two balloons be placed in a calorimeter, one vacuous and the other filled with a gas under pressure, and if these balloons are connected, the thermal effect will remain *nil*. I do not see clearly, therefore, how Mr. Sauveur accounts for the reciprocal evolutions or absorptions of heat at the points A_3 and A_2 by a simple change of distribution of the carbide. These points, A_3 and A_2, moreover, occur in pure iron; hence they cannot depend on the diffusion of a carbide.

From the point of view of mechanics, does the degree of hardness of a material increase with the state of division of its hardest constituent? If 15 per cent. of quartz grains, of 1 cubic centimeter volume, be agglomerated with a suitable cement, and if an attempt be made to file the product, the file will be arrested when it encounters a grain of quartz. If the quartz is in fragments of 1 cubic millimeter, that is to say, in fragments 1000 times smaller than before, the file, it will be evident, will hardly scratch these fragments, though it will probably detach them. If the quartz is in impalpable powder, the proportions being maintained constant, the resistance to the passage of the file will probably be but small. The fact is that hardness, in the mineralogical sense of the term, does not alone come into play. A sufficiently thin plate of glass, resting on a soft support, could be cut without difficulty by a fingernail. Or, to take another example less remote from our subject, it will be seen that, the percentage of carbon being equal (say 1.50 per cent.), cementation-steel, with its relatively thick network of segregated cementite, is harder under the file than is the same steel after forging at a dull-red heat—the cementite after this treatment being much more finely divided.

In the intermediate series of products between steel and white iron it is necessary to reach at least 2.5 per cent. of carbon (37 per cent. of cementite) in order to attain a degree of hardness comparable with that of hardened tool-steels. To put the matter briefly, it seems to me to be evident that, far from increasing the hardness of steel, the minute division of its carbide can only diminish this hardness. I cannot think, therefore, in view of all the reasons which I have set forth, that Mr. Sauveur's theory furnishes the explanation either of the critical points or of hardening.

There is, however, in his theory, an idea which, it appears to me, should not be allowed to be lost sight of, namely, the persistence of the carbide Fe_3C above the point Ac_1. I have thought for some years that at the point Ac_1 there was total dissociation of this compound, because Weyl's electrolytic method isolates from white cast-iron a relatively small proportion only of a definite carbide; and wrongly I trusted to this method for accurately estimating the whole of the carbide. The immediate consequence of this erroneous view was the conclusion that segregated cementite was different from the hard plates of the pearlyte, and represented a variety of martensite very rich in hardening carbon. But this supposition was in no way verified by my subsequent micrographic investigations; and Prof. Ledebur, employing Müller's method, found carbides of practically identical formulas in the entire series of carburized irons from soft steel up to white iron.* It seems probable, therefore, that the segregated cementite and the lamellar carbide of pearlyte are identical.

On the other hand, I have established the presence of fragments of cementite in a steel with 1.24 per cent. of carbon, quenched from a temperature 30° above the point $Ac_{3,2,1}$,† and this fact has been confirmed by the observation of Prof. Arnold‡ and by that of Mr. Sauveur. Thus, under certain conditions, a portion of the cementite can exist above the point $A_{3,2,1}$; and the question arises whether steel above the critical points is not a solution, pure and simple, of Fe_3C in γ iron, *i.e.*, the form of iron which is stable above A_3. This, in fact, is made quite evi-

* *Stahl und Eisen*, vol. xi., p. 294 (April, 1891).

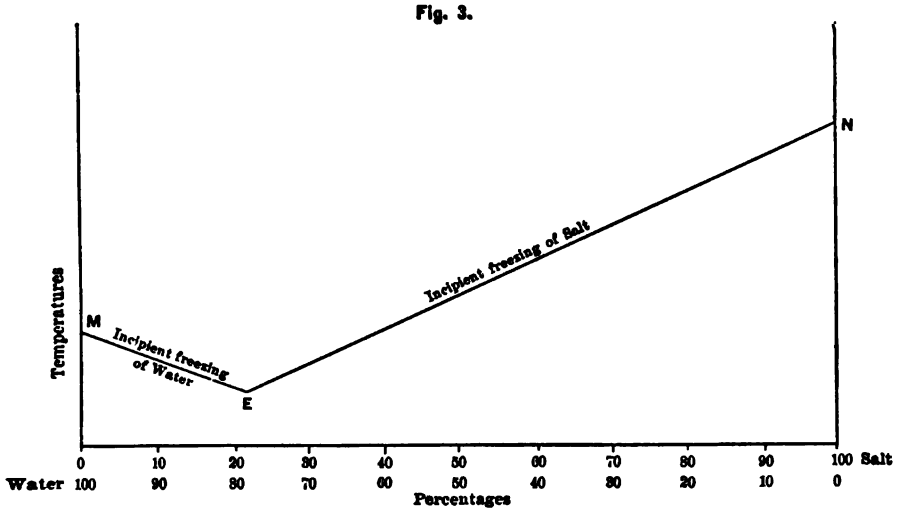
† *Comptes rendus*, vol. cxix., p. 329 (July 30, 1894).

‡ *Proc. Inst. Civ. Eng.*, vol. cxxiii., p. 148–150.

dent by the comparative results of thermal and micrographic investigations.

Usually when two substances, which form neither definite compounds nor isomorphous mixtures, are soluble in each other, it is known that their curve of fusibility (or of solubility, which is the same thing) will be composed of two branches. One of these branches represents the deposition, in a solid state, of one of the substances, and the other branch corresponds to a deposit of the second substance. If it is a question, for instance, of a saline solution (Fig. 3), the branch ME, starting from the point M, corresponds to a deposit of ice in

Fig. 3.



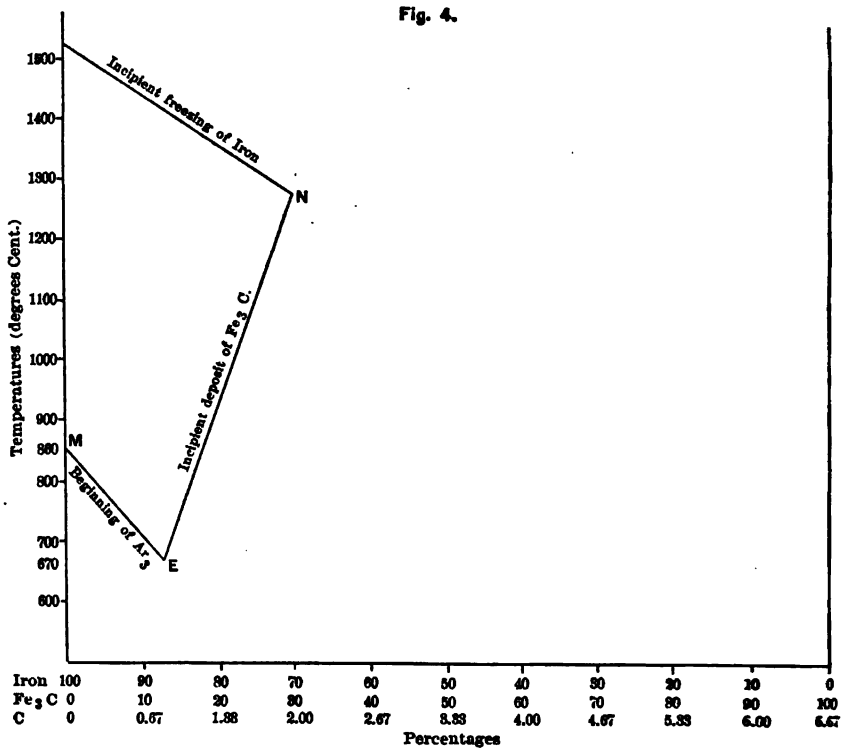
Curve of Solubility in a Saline Solution.

dilute solutions, the point M to the freezing of pure water; the branch NE corresponds to the deposition of salt in a concentrated solution, and starts from the point N, that is, from the fusion of the salt. The point of juncture E is that of solidification of a mixture in such a proportion that the salt and the ice are deposited together at a constant temperature. M. Henri le Chatelier,* from whom these considerations are almost textually borrowed, has shown that alloys behave in the same way, in the case, that is, of those alloys in which the constituent metals are simply dissolved in each other; the point E corresponds, therefore, to the solidification of the *eutectic* alloy of the series.

* *Revue des Sciences pures et appliquées*, 1895, pp. 529–538.

The same thing occurs in the case of steel. There is a curve with two branches (Fig. 4). The branch ME starts from the point Ar_3 in pure iron (860°), and corresponds to the beginning of the segregation of ferrite; that is to say, to the transformation of γ iron into β iron in steels which are not saturated with Fe_3C. The branch NE, which is plotted approximately from the results of my own experiments upon the highly carburized steels, represents the beginning of the segregation of Fe_3C.

Fig. 4.



Curve of Solubility for *Beta*-Iron and Fe_3C.

The point of juncture E (near 670°) is that at which β iron and the carbide Fe_3C are simultaneously deposited at a constant temperature.

This is the point which Prof. Arnold calls the *point of saturation*, attributing it to the formation of a definite compound, $Fe_{24}C$.

The form alone of the curve is sufficient to show that this interpretation is erroneous; but the expression, *point of satu-*

ration, far as it is from representing the views of Prof. Arnold, may perhaps be retained, because it happens to be appropriate to the actual facts for the following reason: It is the concentration at which there is both saturation of γ iron by the carbide Fe_3C, and saturation of carbide Fe_3C by γ iron. At this point the phenomenon which occurs is exactly comparable with the solidification of *eutectic* alloys. In this instance, the eutectic alloy would correspond, to take Prof. Arnold's own figures, to 0.90 per cent. of carbon in the case of pure steels, which is nearly accurate; and the point of saturation is, moreover, lowered by the presence of foreign elements which would be inexplicable by his hypothesis of the existence of a definite compound $Fe_{24}C$.

Furthermore, it is permissible to suppose that the carbide Fe_3C, when dissolved, that is to say in martensite, behaves like a salt, and becomes in part dissociated with the separation of carbon *ions;* and these occur together with molecules which remain intact. According to this hypothesis, the fraction of the carbon which exists as Fe_3C in the solution corresponds to the new variety of carbon which Mr. Hogg thinks it is possible to distinguish by the action of dilute, cold nitric acid.* It is, perhaps, this carbide which characterizes, in annealed steels, the structural constituent which I have called sorbite;† it yields, when Weyl's method is employed, a hydrate of carbon, and when Eggertz's method is applied to hardened steels it yields the brown coloration which is found even in the case of the unsaturated steels, notwithstanding the absence of microstructural cementite. The carbon ions correspond to the carbon which is "missing" when the methods of Weyl, Eggertz or Müller are used.

It is evident, therefore, that the facts may thus be naturally and effectively grouped; but it is necessary to find an explanation for the fact that segregated cementite is not isolated by Weyl's method, as is the cementite derived from pearlyte. The first, which is much less finely divided than the second, ought to offer much more resistance to the reagents. The following explanation appears to me to be not improbable. It is known that segregated cementite exists in abundance in white iron

* *Jour. Iron and Steel Inst.*, 1896, No. II., p. 179.

† *Bull. de la Soc. d'Enc.*, May, 1895.

above the point $A_{3,2,1}$; but it is not magnetic at this temperature, and represents, therefore, an isomer of the lamellar car-

Conditions and Reactions of Carbon in Steel.

Description.			Formula.	Mode of Occurrence in Microstructure.	Results Obtained by the Respective Methods of:		
					Weyl.	Müller.	Eggertz.
Carbon known as cementation carbon.	Cementite.	In fine plates.	Fe_4C.	Pearlyte.	Set free.	Set free.	Soluble, with a brown coloration.
		Segregated.	Fe_3C.	Segregated.	Give a carbohydrate.	Set free, with possibly a certain amount of loss.	Soluble with a brown coloration.
		Dissolved.	Fe_8C.	Martensite. Sorbite. Austenite.			
Carbon known as hardening-carbon.			C ions.	Martensite. Troostite. ?	Partially gasified.	Gasified.	Gasified.

bide of pearlyte. This isomer is certainly stable at the ordinary temperature, when it contains a sufficient proportion of man-

ganese, because ferro-manganese with 25 per cent. of manganese is not magnetic. May it not be the same with the segregated cementite in the white irons and in highly carburized steels?

If we bear in mind that the segregated cementite and the carbide of Mr. Hogg, which is supposed to be dissolved, have the same chemical reactions, we are led to identify them as being the same. Are we not in presence, then, of Mr. Howe's carbide of β iron?

This is a supposition which needs verification. An attempt to verify it may be made by isolating, by Müller's method, a certain quantity of segregated cementite, and ascertaining whether it is magnetic or not. If it is not, the hypothesis becomes a fact; if it is magnetic, we shall still be in doubt, because segregated cementite, when separated from its associations and apart from the conditions which enable it to exist, may have spontaneously assumed the isomeric, magnetic form which is the normal form at the ordinary temperature. If this proves to be the case, some other method of investigation will have to be sought.

For the sake of clearness, I have arranged in the preceding table the conclusions which appear to flow from the preceding considerations as regards the chemical properties of the different forms of carbon other than graphite.

This table, I would repeat, is only the expression of a working hypothesis, fairly accounting for several facts which remain obscure in the chemical reactions of carbon in its various states. This hypothesis admits, with Mr. Sauveur, that there is a solution of carbide Fe_3C, which is, at the same time, the carbide of β iron of Mr. Howe; and as the proportions in which such a carbide can possibly occur by no means afford an explanation of hardening, the *rôle* of the carbide should be that of contributing, concurrently with the free dissolved carbon, to maintain a part of the iron in the β state. I have already, in 1892,* explained, by the existence of a double carbide of iron and chromium in solution, the self-hardening of certain chrome-steels prepared by Mr. Hadfield. In fact, the allotropic theory remains intact, whether iron is retained at the ordinary temper-

* *Jour. Iron and Steel Inst.*, No. II., 1892, p. 127.

ature in the β state by carbon alone, by the carbide Fe_3C alone, or by both together.

The Present Position of the Allotropic Theory.

The allotropic theory, like all living theories, is in constant evolution. Its defenders do not claim to have created all the members which constitute its present form. Their more modest part has been to improve the weapon of research which they have received from their predecessors, and to adapt it to such new facts as have, from time to time, presented themselves. If adaptation remains possible, the theory has proved its vitality.

It appears necessary, therefore, to restate here in some detail an exposition which has been already offered, perhaps too briefly, of the present state of the pending questions.*

Definition of Allotropy.—First of all, what is allotropy? It is well to turn for an answer to Berzelius, who coined the word. Traced to its origin, it will be seen that an allotropic change is a change of properties in an element without change of state (that is to say, with neither the simultaneous passage from the solid to the liquid state, nor from the liquid to the gaseous state, nor, conversely, from the gaseous to the liquid).† In other words, if two or more allotropic forms of a simple body, liquid, solid or gaseous, can coexist under the same conditions of temperature and pressure, they may be distinguished from each other by the possession of different properties. But the progress of science has shown that allotropic changes always correspond to a critical change of energy; that is to say, that the passage of one form to another is effected more or less abruptly, and is attended with either evolution or absorption of heat, and, consequently, the different forms do not possess the same quantity of heat at the same temperature.

A change of energy forcibly entails a change of properties, while outward properties may undergo certain variations without change of energy (for example, the result of annealing is to modify the dimensions and the union of the structural constituents). It is natural, therefore, to trace back the effects to

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* *Jour. Iron and Steel Inst.*, 1894, No. I., pp. 149 and 181; 1896, No. I., pp. 180 and 215.

† *Traité de Chimie; Traduction française de Esslinger et Hoefer*, t. i., pp. 19 à 23, Paris, 1845.

their immediate cause. This has enabled Prof. Roberts-Austen to give the following definition of allotropic change, which appears to be fully adequate: "Allotropy is a change of internal energy occurring in an element at a critical temperature, unaccompanied by a change of state."*

In reality this is not, as we have seen, a new definition; it is simply the old definition of Berzelius (which is also that of Mr. Howe) adapted to the present state of science.

It should be observed that this same definition is equally applicable to isomeric changes if the word "element" be replaced by the word "compound;" and it applies also to changes of state, if the last words "unaccompanied by a change of state" be suppressed. The tendency of all modern physical chemistry is, in fact, to view these changes as being members of the same family, to which different names may be given for the sake of convenience; but they are nevertheless analogous, even if they are not identical.

Now, if a classification has been established among these changes of energy, it is permissible to subdivide the main divisions. It is possible to conceive, for example, that allotropic changes correspond sometimes to a change either in the number of atoms or in their arrangement in the molecule. Allotropy may sometimes be due to a change in the union of the molecules and sometimes to both these changes. Indeed, it has been proposed to make a distinction between physical and chemical allotropy.† But, as there are generally but few means of ascertaining what takes place within the atomic or molecular groups, this distinction is usually speculative, especially in the case of solids.

It is, however, what Prof. Arnold takes to-day as a criterion‡ of allotropy. According to him, chemical allotropy alone exists; physical allotropy is not allotropy. He enunciates the principle that iron presents a case of physical allotropy (though no one knows whether it is or is not so) and consequently has nothing in common with ordinary allotropy. But as the molecular weights of carbon, phosphorus, sulphur, selenium and arsenic

* *Jour. Iron and Steel Inst.*, No. I., 1896, p. 216.

† *Les Théories Modernes de la Chimie*, par Lothar Meyer; *Traduction française*, de A. Bloch et J. Meunier, t. ii., p. 4, Paris, 1889.

‡ *Jour. Iron and Steel Inst.*, 1896, No. I., p. 201.

in their different states are no better known than those of iron, most of the classical examples of allotropy are equally excluded, or their existence is at least called in question.

Mr. Arnold suggests, then, whatever he may say, a new definition very different from that which is at least implicitly accepted by all the world, and this definition rests upon properties which are usually inaccessible to determination.

Allotropic Transformations of Iron.—Prof. Arnold has, in fact, long ago admitted that the point A_2 corresponds to a transformation of plastic iron into crystalline iron.* As Mr. Sauveur has justly observed, these two epithets are about as unfortunately chosen as they could be; for iron which is called "plastic" is crystalline, and iron called "crystalline" is plastic, at least at the critical temperature; but these expressions cannot be considered to convey to the minds of those who use them anything less than the idea of an abrupt change in molecular relations.

In the same way Prof. Arnold† no longer explains the point Ar_3 as due to chemical reaction, but now admits that this point represents a change "analogous to a change of water into ice." But this is all he is asked to do.

Allotropists, without staying to quarrel about words, note this concession. They record the fact that Prof. Arnold is fundamentally in agreement with them, and mark a point gained. They naturally persist in considering the points A_3 and A_2 as signs of the transformations of iron which come under the definition of allotropy, because they are produced in the purest iron as yet obtained "without change of state and with a critical variation of energy."

The points A_3 and A_2 correspond to two independent phenomena. For the want of conclusive proofs, this distinction was not made when the allotropic theory, called the theory of β iron, was originated; but it now claims recognition for two decisive reasons:

1. Because the appearance and disappearance of magnetism coincides with the point A_2 to the exclusion of A_3; and

2. Because A_2 and A_3 remain separated in iron of the

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* *Proc. Inst. Mech. Eng.*, April, 1893, p. 157.

† *Proc. Inst. Civ. Eng.*, vol. cxxiii., p. 263.

very high degree of purity which was prepared by Prof. Roberts-Austen.*

I am aware that the identification of the point Ar_2 with the appearance of magnetism seems to be in contradiction with certain experimental results. It is necessary, therefore, to define carefully again in this place what is understood by the "appearance of magnetism." Since this phenomenon is, in reality, progressive, and extends over a somewhat large range of temperature, its position on the scale of temperatures, if it is to be determined by a certain induction-value fixed beforehand, will necessarily depend on the choice of this value. In the same way two equally accurate observers might find some widely divergent numbers for the elastic limit of a metal, if they had not agreed beforehand as to the amount of permanent elongation which marks this elastic limit. In the main, the divergences only exist in conventional usage.

The rapidity of cooling, the intensity of the field and the kind of pyrometer are also causes of other apparent disagreements which are explained by the differences in the experimental conditions. And so, to avoid confusion, it appeared to be necessary to record simultaneously, in a single experiment, the cooling-curve and the progressive change of magnetism.

This was what I did; and the results, lately confirmed by M. Curie's far more precise and extended experiments,† have left no doubt in my mind as to the coincidence of the appearance of magnetism with Ar_2. Moreover, the particular steels which have no point A_3, nevertheless lose their magnetism when heated to redness.

If, however, any uncertainty should still survive, the cooling-curve of pure iron obtained by Prof. Roberts-Austen alone suffices to separate Ar_2 from Ar_3.

I insist on this distinction, which I have already indicated elsewhere, because it is fundamental, and because sufficient attention has not always been paid to it. It necessarily compels us to distinguish three allotropic varieties of iron, α below A_2, β between A_2 and A_3, and γ above A_3, while originally the two varieties, β and γ, were provisionally comprised under the name

* *Proc. Inst. Mech. Eng.*, April, 1895, p. 244.

† *Thesis*, Gauthier-Villars, Paris, 1895.

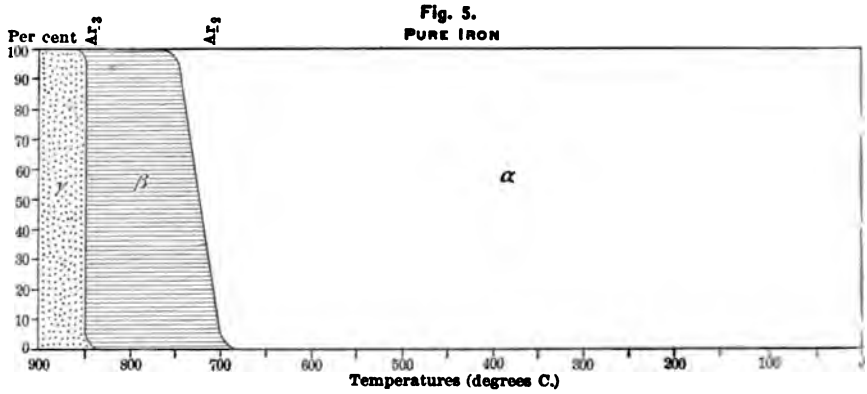
of β iron. We ought even, at present, to consider it possible, in view of the experiments of Dr. Ball* and M. Curie, that it may still be necessary to subdivide γ iron.

A small difficulty now presents itself which had not hitherto been raised. Mr. Sauveur does not seem to be quite certain that the points A_3 and A_2 of the iron are included in the double or triple points $A_{3,2}$ or $A_{3,2,1}$ of the steels. To me, however, the existence of such a doubt seems impossible for the following reasons:

1. When conducting experiments on the cooling of steels with an increasing percentage of carbon, it is easy to follow the displacement and running together of the critical points.

2. Prof. Arnold† has calculated the numbers of calories released at the point Ar_1, at the same time deducting the sum of

Fig. 5.



Relative Proportions of Each Variety of Iron Occurring in Pure Iron During Slow Cooling.

$Ar_3 + Ar_2$ from the supposed triple points, and has found that these calories were proportional to the percentage of carbon up to 0.90 per cent. It is true that the method of calculation was not very exact; but the results, though approximate, well justify the hypothesis.

3. The appearance of magnetism, which is characteristic of the point Ar_2, occurs in points $Ar_{3,2}$ or $Ar_{3,2,1}$, which, it is admitted, include Ar_2.‡

4. The curves of thermo-electric power§ and of electrical re-

* *Jour. Iron and Steel Inst.*, 1890, No. I., p. 85.

† *Proc. Inst. Civ. Eng.*, vol. cxxiii., p. 152.

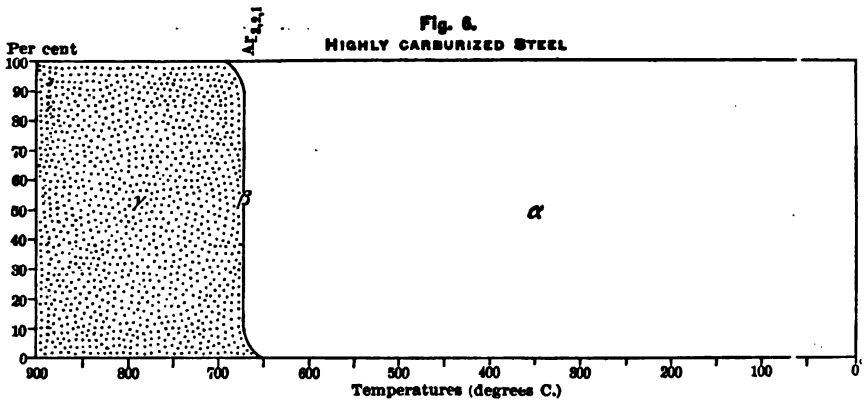
‡ Hopkinson, *Proc. Roy. Soc.*, vol. xlviii., 1890, p. 442.

§ Osmond, *Transformations of Iron and Carbon*, pl. vii. and viii.

sistance,* considered in relation to temperature, reveal the existence at the point A_2, when it stands apart, of some typical characteristics which are gathered together in the united points $A_{3,2,1}$ in highly carburized steels.

These proofs seem conclusive. If they be accepted as such, one can represent by special diagrams the relative proportions of each variety of iron which exist during the course of slow cooling, at any given temperature and for any given variety of steel. For carbon-steels, these diagrams may be divided into three types.

In pure iron, if Ar_3 is suddenly produced at 855° C., and Ar_2, progressively, between 750° and 700°, then the diagram would be such as is shown in Fig. 5, which explains itself.



Relative Proportions of Each Variety of Iron Occurring in Highly Carburized Steel During Slow Cooling.

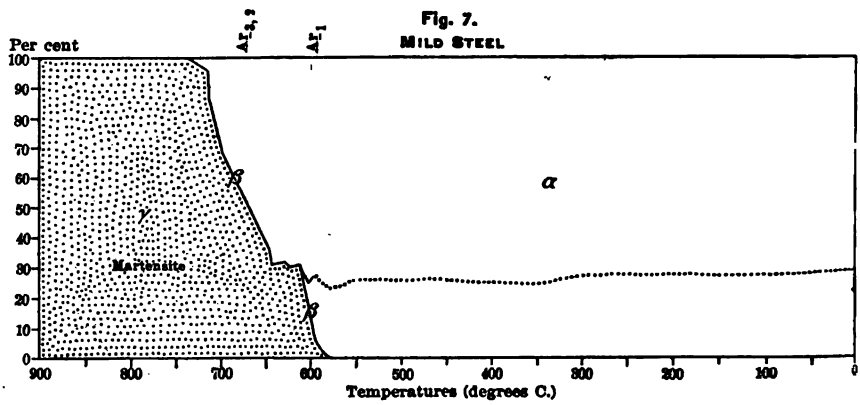
In a highly carburized steel, the three points are joined into one, at 675° (for example); and the conditions are represented by Fig. 6. β iron becomes α iron directly it is formed.

In a variety of steel presenting two critical points, things are more complicated. Take, for example, the steel so closely examined by Howe and Sauveur. The diagram which gives the relative proportions of the three forms of iron is evidently the same as the diagram showing microstructural composition (Fig. 7). In this case there is, during the critical interval, a mixture of martensite, the proportion of which is progressively decreasing, and ferrite, the proportion of which is naturally varying in

* H. le Chatelier, *Comptes rendus*, vol. cx., p. 283.

a contrary sense. Each of these constituents has its own critical points. The point $Ar_{3,2}$ does not belong to the whole mass, but to ferrite alone. In the martensite, which finally becomes a hard steel with about 0.80 per cent. of carbon, the progressive concentration of the carbon delays the transformations of the iron, just as in a true highly carburized steel; and the point which is called (for the mass) Ar_1 is really the point $Ar_{3,2,1}$ of the martensite.

There is no γ, β and α martensite; there is only γ martensite during slow cooling. And this answers Mr. Sauveur's objection; for he asks why low-carbon steel, quenched between $Ar_{3,2}$ and Ar_1, does, in fact, harden. In the martensite the



Relative Proportions of Each Variety of Iron Occurring During Slow Cooling in Steel with two Critical Points.

point $Ar_{3,2}$ has not begun, whilst it has ended in the ferrite. Martensite hardens like all carburized steels, and consequently imparts some hardness to the mixture. It is true that the notation in this case is not very convenient. It would be necessary, in the example under consideration, where the final proportion of martensite is 23 per cent., to replace $Ar_{3,2}$ by $\frac{77}{100} Ar_{3,2}$, and Ar_1 by the sum $\left(Ar_1 + \frac{23}{100} Ar_{3,2}\right)$. But this question of critical points in soft steels was not completely cleared up until the recent investigations in metallography had been made.

Permanence of Allotropic Forms.—The polymorphism of iron once proved, it remains to be seen whether the forms β and γ can be maintained, either wholly or in part, at the ordinary temperature. Mr. Hadfield has pointed out that allotropic states are often unstable;* but this is purely an abstract objection. It is quite certain that each distinct form is only naturally in equilibrium between certain limits of temperature. It is a question to be solved whether it is possible to maintain the form beyond these limits, and, if so, how. If plastic sulphur and ozone do not long retain their identity, the diamond and red phosphorus preserve theirs indefinitely. Pyrophoric iron, reduced by hydrogen, appears to preserve its form, provided it is not exposed to the air.

Mr. Hadfield himself, while saying that the sub-carbide of Mr. Arnold is "extraordinarily sensitive to change,"† now appears to think it quite natural that this carbide should exist at the ordinary temperature. I do not see how, in ordinary fairness, a privilege can be refused to β iron which has been generously accorded to the so-called carbide $Fe_{34}C$. But it is not a question of privilege in the particular case before us. We have to see, first, if β and γ iron can be so maintained, and, further, if they really are so maintained.

Generally, in order to maintain a chemical system beyond its natural limits of equilibrium, it is necessary to introduce some passive resistance into the system.

Of the nature of these resistances we know but little. But we know, for example, that a mixture of oxygen and hydrogen can exist indefinitely at the ordinary temperature, while its stable form at this temperature is water; and sudden cooling is usually a means of traversing the range of temperature in which reaction is possible, in order to reach a temperature at which the introduction of passive resistances impedes or stops reaction.

We know, moreover, that dissolved bodies exercise on the solvents what is known as "osmotic pressure," and that osmotic pressure lowers the freezing- and volatilization-points of the solvent.

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* *Trans.*, xxiii., 154, 155.

† *Jour. Iron and Steel Inst.*, 1896, No. I., p. 197.

We know, further, that pressure (in the ordinary sense of the word) also displaces the freezing- and volatilization-points, lowering them or raising them, according to the sign of the pressure, and that of the change of volume which accompanies the change of state. The same is necessarily true for allotropic changes, as Mallard and H. le Chatelier have directly demonstrated by lowering more than 100°, under a pressure of 3000 atmospheres, the isomeric change of iodide of silver.* Spring and Van't Hoff have shown that the change-point of copper-calcium-acetate is lowered 40° by the application of a pressure of 6000 atmospheres; and Reicher finds that sulphur has its change-point raised 0.6° by a pressure of 11.8 atmospheres. During rapid cooling, similar effects of tension and compression are produced, and Prof. Roberts-Austen has established the fact that there is a difference of more than 250° between the temperature of recalescence at the surface, and at the center, of a little ingot of quenched steel.†

What parts are played by these general factors in the case of steel?

Experiment shows that at a red heat carbon diffuses itself in steel and delays the occurrence of the points Ar_3 and Ar_2, causing them to coincide with Ar_1. Experiment also teaches us that manganese, nickel and copper (that is to say, those elements of which the atomic volume is smaller than that of iron), and certain dissolved double carbides of iron and manganese [$(Fe,Mn)_3C$], of iron and chromium [$(Fe,Cr)_3C$], of iron and tungsten (?), also lower the points of transformation, sometimes even to below the ordinary temperature. Such are the clear manifestations of the existence of osmotic pressure; and if it should seem surprising that such pressures should be referred to in connection with solids, it is only necessary to bear in mind that there is no sharp transition between solids and liquids, and to cite the beautiful experiments by the aid of which Prof. Roberts-Austen, continuing the work of Graham, has shown that gold will diffuse in *solid* lead near the ordinary temperature.‡

As regards external pressures, they are not purposely called

* *Comptes rendus*, vol. xcix., p. 157, July 21, 1884.

† *Proc. Inst. Mech. Eng.*, Oct., 1891, pp. 543 *et seq.*

‡ *Trans. Roy. Soc.*, vol. clxxxvii. (1896), pp. 417–432.

into play except in Clemandot's process of hardening steel. They intervene, however, during rolling, when it is continued below redness, and still more (though the fact may be overlooked) in quenching steel. M. André le Chatelier has called attention (and it is well that he has done so) to the importance of the changes of volume which accompany the transformations of iron.* Each allotropic variety has its own co-efficient of expansion; the more the critical points are lowered the greater will be the abrupt changes of volume; and, according to the calculations of M. André le Chatelier, they may cause tensional or compressional stresses which may amount to 187 kg. per square mm. if the changes take place at 300°, whence the frequent fractures caused by quenching steel. I cannot recognize in this stress a satisfactory explanation of the hardness of quenched steel. But the fact does exist; and it is natural that the considerable tensions produced in a body which is practically incompressible, at too low a temperature for spontaneous annealing to efface the strains, should determine the permanent preservation of an allotropic form of iron which would be unstable at the ordinary atmospheric pressure and temperature. As, however, the pressures are engendered by the changes themselves, a change cannot be arrested if it has not already been partly accomplished. Thus the vaporization of water in a closed space will be stopped when the pressure attains a certain value which is a function of the temperature; but the pressure which limits the vaporization is caused by the vaporization itself, and this presupposes that a portion of the water had previously been changed. This example well shows the theoretical possibility, and probably the necessity, of the simultaneous presence of at least two forms of iron in iron-metals in which changes have been delayed. But the complete solution of the problem demands investigations similar to those made by Regnault in the case of steam. It is necessary to know the laws of the expansion of the three varieties of iron and their pressure of transformation for all temperatures, and it would seem, moreover, that certain portions of the necessary researches do not admit of investigation by direct experiment.

In fact, we know thoroughly only two extreme cases. One is, that if pure iron be slowly cooled, all the changes are com-

* *Bull. de la Soc. d'Enc.*, vol. x. (1895), p. 1342.

pletely effected in it at the normal temperatures: the whole of the iron remains in the α state.

The other case is presented by manganese-steel with 12 or 13 per cent. of manganese and with about 1 per cent. of carbon, and by nickel-steel with about 25 per cent. of nickel. In these varieties of metal no change occurs during normal cooling; the iron in them remains wholly in the γ state. This proposition is a mere truism. It might as well be said that water remains water, even below zero, so long as its heat of liquefaction has not been evolved.

As regards the intermediate cases, which constitute the great majority, and especially include the operation of quenching, difficulties surge up. The facts which are known may be grouped as follows:

1. The quenching of steel obliterates the changes, because tempering is accompanied with an abnormal evolution of heat and a spontaneous acceleration of the speed with which the temperature of the metal rises; but they are not completely suppressed, because quenched carbon-steels are magnetic, thus differing from manganese- or nickel-steels of suitable composition.

2. In all cases in which, from whatever cause, the critical points are lowered below about 350° to 400° C., varieties of steel are obtained which are more or less hard, magnetic and polar-magnetic.* As, however, whenever it is possible to trace the cooling, an evolution of heat is established, and as, moreover, the metals under consideration are magnetic, it is certain that changes are at least partially effected in them; but there are strong reasons for the belief that these changes remain incomplete, because the properties the metals finally possess differ considerably from those of soft iron.

3. In those steels in which the changes are effected below 350° to 400° C., we see that the hardness either begins or increases with the point $Ar_{3,2}$.†

4. In the case of certain nickel-steels with a suitable percentage of nickel, in which the changes begin near the ordinary temperature, the changes can be stopped at will by stopping the cooling at any determined temperature.

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* *Jour. Iron and Steel Inst.*, 1896, No. I., pp. 180–187.

† *Comptes rendus*, vol. cxviii., p. 532.

But here theory intervenes in order to generalize these experimental results, and the theory may be embodied in the following proposition:

When the causes, whatever they be, which tend to lower the change-points are neither sufficiently powerful to completely suppress them nor sufficiently weak to allow the changes to be completely effected, above 350° to 400° C., then the changes will remain incomplete and will leave in the metal a mixture in variable proportions of α iron, β iron and γ iron. As α iron is soft and γ iron does not possess a high degree of mineralogical hardness (although it cannot be easily machined), we attribute the hardness to β iron, or to a combination of β iron with one of the other varieties.

This proposition is rendered very probable by the known facts; but it is not rigorously demonstrated, and the last hypothesis of allotropists consists in admitting it.*

If we accept it, the knowledge of the respective proportions of α, β and γ iron present in a mixture will give the first general idea of the most characteristic properties of the metal. It is naturally not a question of predicting *all* the possible properties in detail. In the first place, the anticipations thus formed would not be applicable except to a chemically homogeneous mixture—that is to say, to each constituent individually; and if the metal contains many constituents, as often happens, the knowledge of each of them, even if it be complete, will not always enable the properties of the whole to be defined with precision. It is also necessary to reckon with certain variations of structure, apart from the molecular state of the iron. For example, there are joints in the ferrite which are more or less pronounced, according to the temperature and the time occupied by annealing; there is the state of division of the segregated cementite, etc. Furthermore, in the usual industrial products, interposed foreign matters may be met with, such as oxides, sulphides, phosphides, arsenides and slag, not to speak of blow-holes and of "pipings," which have become more or less welded. These are so many disturbing causes which should be studied separately, and which, as re-

* It will be understood that it is necessary also to admit that iron is an element, and to admit provisionally that the change which appears to take place at 1300° C. does not intervene in practice.

gards the purely theoretical anticipations, will lead to grave errors if they are not taken into account.

There remain, however, some properties which appear to be less troubled by these disturbing causes than are the others, and these are notably mineralogical hardness, electrical resistance and magnetism.

Fig. 8.

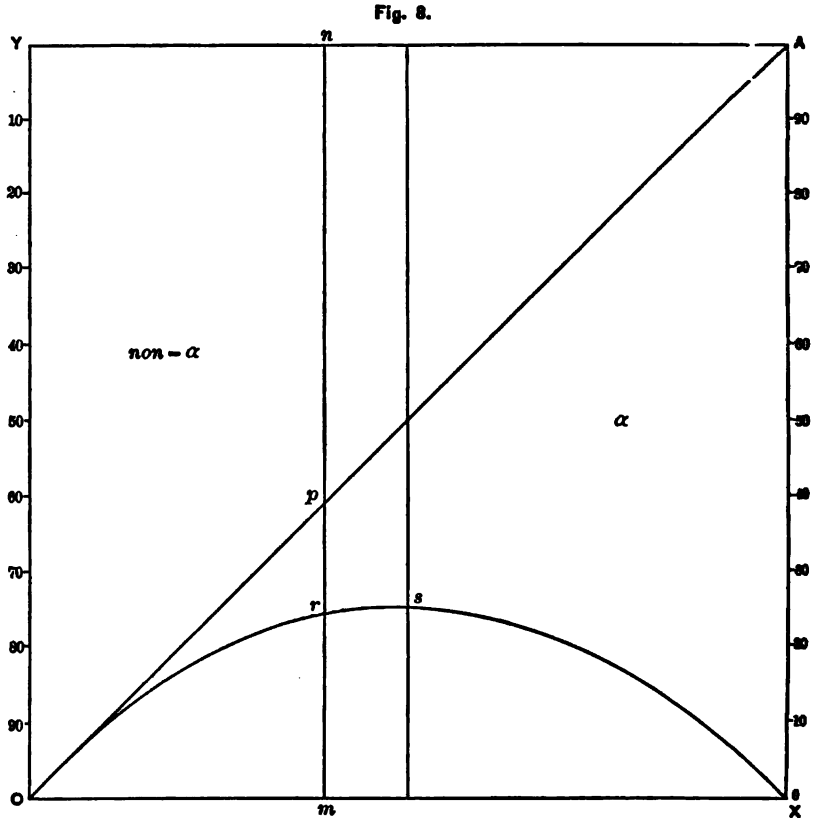

Diagram Illustrating the Magnetism of a Mixture of *Alpha* with other Iron.

Let us take, for example, magnetism and examine its theoretical aspects with relation to the allotropic forms of iron.

The α form of iron is magnetic. But, unless special precautions are taken, it is not polar-magnetic—that is to say, the bar ceases to be a magnet and the molecules resume their original positions when the circuit is interrupted.

β and γ iron are non-magnetic, and for this reason we can

here group them under the name of non-α iron—a name which Mr. Howe has already used.

It is easy to see that, in a molecular mixture of α and non-α iron, the latter can act as a passive resistance, forming an obstacle to the polarization of the iron, and that this will retain the polarization once it is obtained. In other words, the presence of the inert molecules of non-α iron would be a necessary and sufficient cause for what was formerly called "coercive force." In the same way the "load-stone" (Fe_3O_4) is formed of a magnetic body (FeO) and a non-magnetic body (Fe_2O_3).

Let OAXY (Fig. 8) be a square, of which the side represents 100 parts. An ordinate mn, cutting the diagonal OA at p, shows a mixture of mp per cent. of α iron and of np per cent. of non-α iron; and, if we employ a current strong enough to polarize the whole of the α iron present, the diagonal OA can be taken as a measure of the total magnetism.

Now, let it be admitted that the proportion of the iron which will remain polarized after the interruption of the current—that is to say, the residual magnetism, is proportional to the amount of non-α iron in the mixture. Let $mr = y$ be the amount of the permanent magnetism which corresponds to the value

$$mp = mo = x$$

of total magnetism.

We have, by hypothesis,

$$y = kx\left(\frac{100 - x}{100}\right) = k\left(x - \frac{x^2}{100}\right),$$

k being a constant not greater than unity. It is the equation of a parabola, such as $OrsX$, of which the axis is parallel to OY. The residual magnetism is *nil* when the whole of the iron is in the α state; it passes to a maximum in a mixture of equal parts of α iron and non-α iron, and evidently becomes *nil* when the α iron disappears.

All this appears to be thoroughly in accordance with the facts. The induced magnetism is at a maximum in the pure iron, and diminishes progressively in proportion as the carbon, the manganese or the nickel increases.

At the same time the permanent magnetism at first increases proportionately as the induced magnetism diminishes; it passes

through its maximum in hardened steels, containing about 0.90 per cent. of carbon, then decreases (Austenite) and disappears in manganese- and nickel-steels of suitable composition.

Physicists may object that the total magnetism of hardened steels appears to approximate more and more closely to that of soft iron when the intensity of the magnetic field is indefinitely increased, and Mr. Hadfield will doubtless say that his manganese-steel becomes magnetic in a field which is sufficiently intense.* This merely shows that a very strong electric current sets up a sort of temporary strain, and, whatever may be the original state of the molecules, tends to modify the relative proportions of α iron and of non-α iron, so as to approximate them to the same state of equilibrium. It is, in fact, known that magnetization does change the dimensions of the bars, and, on the other hand, that strain which increases the coercive force of annealed steel may magnetize non-magnetic steels. Mr. Hadfield has pointed out that certain manganese-steels which are not magnetic in mass give magnetic shavings;† and, for my own part, I have observed that, if the surface of non-magnetic steel be scratched with a diamond, the powder is invariably attracted by a magnet. Furthermore, the experiments of M. Curie‡ show that the total magnetism of soft iron, which is practically constant between 0° and 500° C. in a field which is lower than 300 units, diminishes rapidly with increase of temperature in a field of 1000 units. Magnetization seems here to produce non-α iron.

These considerations, which in the present state of our knowledge are necessarily somewhat vague, will at least serve to show that the magnetic phenomena come without difficulty into line with the general scheme of allotropy.

The Part Played by Carbon.—In the preceding statements it has been a question of the methods generally which enable iron to be preserved in one of its allotropic states. We think, in fact, that every foreign element may, in its relations with steel, present a specific character, and demands for this reason special study; but these more limited theories ought only to be considered as branches of the general theory. Under these circumstances, allotropists have relegated carbon to a secondary

* *Jour. Iron and Steel Inst.*, 1894, No. I., p. 158.
† *Ibid.*, p. 157. ‡ *Loc. cit.*

place, and, for doing so, they have been very unjustly reproached. Directly any experimenter turns his attention to the influence of carbon, the results, whatever they may be, are claimed by Mr. Hadfield as presenting so many arguments against the partisans of allotropy, as if the latter had not contributed anything to our knowledge of the relations of carbon and iron. It is easy to show victoriously that carbon plays a considerable part in the science and practice of metallurgy; in fact, no one doubts it. It is not a question of knowing whether carbon acts, but why it acts, and how it acts.

Allotropists think (and believe they have proved) that its action is not essentially different from that of many other bodies. Electrolytic iron is as hard as quenched steel; steel with 15 per cent. of nickel, and with 0.20 or even with 0.10 per cent. of carbon, is not in practice easily workable after slow cooling. But carbon is a body which presents itself, so to speak, in two different aspects, because it has different relations with different forms of iron. In the presence of α iron it forms a definite carbide, independent of the mass which contains it; in the presence of γ iron it is dissolved, either as free carbon or as a carbide, like nickel or manganese. If cement-carbon alone existed, we should only know the series of annealed steels which are progressively connected with white cast-iron; quenching would produce no effect. If hardening-carbon existed alone, the series of carburized steels would be comparable with the nickel-steel series, but with the quantitative difference that the dissolved carbon, owing to its small atomic volume, is, like the hydrogen in electrolytic iron, the most powerful known modifier of iron. For the increasing percentage of iron there would be, at first, a slight progressive increase of hardness, then the self-hardening steels, then non-magnetic metals, comparable with the manganese-steels. Here, again, quenching would still be either useless or an almost useless operation. The two words "hardening" and "quenching" are so closely identified with each other that many, even careful workers, cannot distinguish the different ideas which they represent. In reality, immersion is only a particular means, applicable in a particular case, for rendering steel hard. It acts by delaying the carbon-change, and, as a consequence, delays the iron-change, also, to such a sufficiently low temperature that the changes will remain in-

complete. This happens spontaneously during the slow cooling of self-hardened steels. The manganese, chromium and tungsten perform the same office as immersion.

But if, in the progress of siderurgy, carbon has been deposed from its position of absolute monarch, and has received that of prime minister, it is not the fault of those only who confess themselves to be allotropists. A leader of the Conservative party, Mr. Hadfield, by the unexpected and remarkable discovery of manganese-steel, proved himself to be one of the most active insurgents in the revolution which he now tries to quell.

Reply to the Objections of Mr. Sauveur and to Some Others.

The general exposition just given of the present state of the allotropic theory will enable the writer to reply in a few words to such other objections as have not been directly answered.

These objections may be divided into four groups:

1. Objections which in principle are well based, but which relate to the primitive and, to-day, antiquated form of the allotropic theory.
2. Objections which are reasonable, but which are beside the question.
3. Objections which may be called sentimental.
4. Illogical objections.

First Group.

(1) *If hard and non-magnetic β iron is the cause of the properties of quenched carbon-steels and of the properties of manganese-steels, the latter, which are non-magnetic, should contain β iron only, while quenched carbon-steels, which are strongly magnetic, can only contain it as a part of their mass. Consequently, manganese-steels should be at least as hard as quenched carbon-steels; and the fact is, they are much less hard.* (Sauveur, Howe, etc.)

The manganese-steels, which undergo no transformations, contain only iron which may be defined as of the γ form. The quenched carbon-steels which have undergone partial transformations owe their hardness to β iron.

(2) *If the absence of magnetic properties is owing to the presence of hard β iron, when a nickel-steel, which is non-magnetic at the ordinary temperature, becomes so (after cooling to —40° C., for instance),*

the metal which has become magnetic should be less hard than the same metal in the non-magnetic form. The contrary, however, is found to be the case. (A. le Chatelier.)*

The non-magnetic metal contains γ iron only; the transformation at about —40° remains incomplete, with the formation of β and α iron.

Second Group.

(3) *A manganese-steel with about* 12 *per cent. of manganese has been treated by cementation. After this treatment it has become magnetic and has retained practically the same hardness; therefore, the magnetism is not connected with the hardness.* (Hadfield.)†

It follows from the experiments of Mr. Stead‡ and from my own§ that the steel in question, which has become magnetic, is built up of three constituents:

1. A hard, non-magnetic carbide, $(Fe,Mn)_3C$.
2. A less hard constituent around this carbide.
3. The remainder, a soft constituent.

In the mixture, the hardness and the magnetism are not referable to the same constituents.

(4) *The same manganese-steel, simply annealed for a long time and cooled very slowly, becomes magnetic without its composition being changed.* (Hadfield.)‖

It is probable that the carbide $(Fe,Mn)_3C$ is, in this case also, liquated, for it sensibly liquates even during cooling in air. The percentage of manganese, apart from the carbide, is not sufficient to hinder the changes, and the metal has consequently become magnetic. This affords a further proof that the carbon contributes to give manganese-steel its ordinary properties; but the fact was not questioned. Moreover, it has not been shown that the cooled metal was homogeneous.

(5) *The same manganese-steel, decarburized by prolonged annealing in oxide of iron and cooled very slowly, becomes magnetic and soft without the percentage of manganese in it being perceptibly changed. After decarburization it can be forged or hardened without becoming non-magnetic again.* (Hadfield.)¶

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* *Jour. Iron and Steel Inst.*, 1894, No. I., pp. 207–210.
† *Ibid.*, 1894, No. I., pp. 156–180.
‡ *Ibid.*, 1894, No. I., pp. 191–196.
§ *Ibid.*, 1894, No. I., p. 218.
‖ *Ibid.*, 1894, No. I., pp. 156–180. ¶ *Loc. cit.*

The percentage of manganese, in the absence of carbon, is not sufficient to prevent the transformations during cooling, either slow or rapid; and the metal is magnetic in consequence. It is to be remarked that it is not hard. This experiment proves still further that carbon contributes in giving manganese-steel its ordinary properties. But that is not the question. Besides, the cooled metal is probably not homogeneous.

(6) *A steel containing* 3.50 *per cent. of manganese and* 0.54 *of carbon scratches glass; a steel containing* 3.50 *per cent. of manganese and only* 0.10 *of carbon can be filed easily.* (Hadfield.)*

This proves equally the importance of the carbon; but it does not prove any the less the importance of manganese. Mr. Hadfield has been kind enough to send me a sample of the second metal named. In it the points of transformation of the iron during slow cooling are really very much lowered by the manganese alone, but not sufficiently to enable the metal to be a self-hardening one. Its homogeneity is doubtful.

(7) *Soft iron cannot be hardened by quenching.* (Hadfield.)†

It has not been proved that carbon-free iron in very small specimens, quenched at a temperature above 900° in an extremely cold liquid, will not harden by quenching. But, if it were proven, the allotropic theory would not be impugned. To obtain marked hardening it is necessary, and it is sufficient, that the temperatures of the transformation of iron should be lowered below 400° at least. In every known case this result is obtained by the concurrence of several factors. It is by no means necessary that one of these factors, namely, the speed of cooling, should alone suffice.

Third Group.

(8) *There is no evidence that quenched high-carbon steels contain a iron. Prof. Arnold, who has "so minutely examined and read the structure of steel," does not see it under an enlargement of* 800 *diameters.* (Hadfield.)‡

Prof. Arnold will hardly claim to have seen the molecules of steel by an enlargement of 800 diameters.

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* *Proc. Inst. Civ. Eng.*, vol. cxxiii., p. 198.

† *Jour. Iron and Steel Inst.*, 1896, No. I., pp. 192–195.

‡ *Ibid.*, 1896, No. I., p. 196.

(9) *One cannot see β iron.* (Hadfield.)*

Neither can the carbide $Fe_{24}C$ be seen. The quenched steel is seen, the properties of which differ from those of annealed steel, and we all seek to discover why.

Fourth Group.

(10) *Bodies with a large atomic volume, notably phosphorus, do not maintain iron in the β state. They ought to make it softer and less fragile.* (Arnold.)†

This argument supposes that no other agents than β iron are susceptible of rendering iron harder or more fragile. The reasoning is exactly comparable to the following: "A man has been vaccinated against small-pox; therefore nothing will ever make him ill."

Objections of this kind abound in the polemics of Mr. Arnold, and this example would appear to be sufficient.

To sum up the question, the allotropists have established, with the useful assistance of their opponents, the existence of two allotropic states of iron and the permanence of these states in certain determined cases. They have given a rational classification of the action of foreign elements connecting the carburized steels to the manganese- and nickel-steels, showing that steel above its critical points is a solution, and furnishing in answer to all objections replies which are at least plausible. They see no reason why their ideas should not be reconciled with the dissolved carbide of M. Sauveur, the carbide of β iron of Mr. Howe, the double carbides of iron and manganese of Mr. Hadfield, and the molecular stresses of M. André le Chatelier.

They regret not having drawn a complete picture, needing no finishing touches, of all the observed phenomena in this branch of the metallurgy of iron. But such a work was not easy to execute. It has not been completed sooner probably because the state of science generally still renders it impossible to effect the solution of a problem which is not the direct outcome of one already solved.

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* *Jour. Iron and Steel Inst.*, 1894, No. I., p. 176. † *Ibid.*, 1894, No. I., p. 203.

HENRY M. HOWE, Boston, Mass.: Besides his most important microscopic results, Mr. Sauveur presents very clearly, and with rare impartiality, a most complex and difficult subject, as to which those of us who dare may more than once have to modify our judgment.

The problem before us in its very nature demands unusually close and patient reasoning. The case is not unlike what that of the discovery of Neptune from the perturbations of Uranus would have been, if Neptune, in addition to being previously unsuspected, had also been invisible; for, unfortunately, we have as yet no means of directly detecting the allotropic forms of iron, the existence of which we have at least the very strongest reasons for suspecting, as no reasonable man with knowledge of the facts can deny.

So long as we had little to explain besides (1) the loss of the magnetic properties at redness, and the facts (2) that the hardening power of steel increases nearly proportionately to its carbon-content, and (3) that the condition of the carbon in hardened steel differs from that in unhardened steel, the carbon-theory easily held the field. For though the magnetic phenomena were explained on neither theory, and though it was possible to explain the remaining facts, (2) and (3), on the allotropic theory, by maintaining that the presence of carbon catalytically restrained the hard high-temperature β iron from changing, even during quick cooling, back to the normal soft α iron of annealed steel; yet, as Mr. Sauveur well puts it, this explanation lacked cogency. It seemed forced, and few entertained it seriously. The carbon-theory, however, explained (2) and (3) very readily.

So far, then, while one theory explained as much as the other, the greater simplicity and directness of the carbon-theory, and the fact that, while the change in the condition of carbon was readily verified by chemical analysis, the supposed allotropic change in the condition of iron could not be, properly raised a very strong presumption in favor of the carbon-theory, which thus appealed irresistibly to most minds.

But, of three remarkable phenomena revealed by further investigation, all completely in accord with the allotropic theory, one was wholly inexplicable, and the two others explicable only with difficulty, on the carbon-theory. And, further, it was

found that the allotropic theory explained the magnetic phenomena most lucidly, while no corresponding explanation of them on the carbon-theory has, so far as I know, been offered. There have been re-statements, but I have seen nothing worthy the name of explanation.

These three phenomena are the following. For ease of exposition I do not observe chronological order:

A. Iron completely free from carbon was found to undergo two spontaneous retardations during slow cooling at the specific temperatures at which the magnetic properties are recovered.* Such retardations imply that heat is spontaneously evolved, and thus imply some change of internal energy. This change in a pure chemical element is very strong evidence of allotropy, especially as it accompanies the recovery of the magnetic properties. It is hard to resist the conclusion that these retardations represent an allotropic change from a high-temperature, unmagnetizable metal, β iron, to a low-temperature, magnetizable one, α iron.

If these phenomena occurred without change of state in a chemical compound, they would, I think, be held without question to prove isomerism. I cannot see why, occurring in a pure element, they are not equally strong evidence of allotropy.

Here let us impress on our memory the fact that the upper retardations are thus identified with allotropy. The lower retardation (the recalescence, Ar_1), on the other hand, is identified with change in the condition of carbon, by the fact that the strength of this retardation increases with the percentage of carbon within limits, and by the fact which I detail in C, that the change in the condition of carbon has been shown to occur at Ar_1, but not at the upper retardations.

B. Iron almost chemically pure, containing only 0.022 per cent. of carbon and 0.121 per cent. of all other elements, can be much strengthened and greatly embrittled, besides having its elastic limit greatly raised, by sufficiently rapid cooling.† The carbon theory fails to explain this readily, because there is

* Roberts-Austen, *Proc. Inst. Mechanical Eng.*, April 26, 1895, p. 244; "Third Report to the Alloys Research Committee."

† The writer: Steel containing

Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.	Copper.
.022	trace.	0	.007	.014	0.10

(the carbon and manganese are both determined by Booth, Garrett and Blair, the former by combustion), in the form of small bars, and, in one case in the form of

too little carbon present, and for that matter too little of anything but iron, to account readily for these effects. We hesitate to refer them to stress, because, so far as we know, no other metal is affected to such a degree, though other metals should, under like conditions, receive great stress. It has been shown that, if anything, these effects are probably induced in spite of stress, rather than because of stress; for on drilling out the interior of hardened bars of this metal, and also by removing

wire, was quenched from about 900° C. in iced brine at from minus 5° to minus 7° C. The wire was quenched in brine at —14½° C. The results are condensed in the following table:

No.	Size. Inches.	Bored or Not.	Size of Holes Bored. Inches.	Tenacity. Pounds per Square Inch.	Elastic Limit. Pounds per Square Inch.	Elongation. Per Cent. in 1 Inch.	Contraction of Area. Per Cent.	Treatment.
6	.162 x .174	Not.		48,580	23,050	44.	72.3	Cooled slowly from 900°C.
10	.187 x .204	Yes.	.063 x .062	49,890				Cooled slowly from 900°C.
2	.176 x .181	Not.		52,040	30,400	43.	77.8	Quenched in water from 900° C.
9	.168 x .198	Not.		67,400	42,820	28.	58.56	Quenched in iced brine from 900° C.
3	.177 x .186	Yes.	.078 x .074	70,690				Quenched in iced brine from 900° C.
7	.178 x .197	Yes.	.086 x .085	60,190				Quenched in iced brine from 900° C.
8	.179 x .195	Yes.	.087 x .088	62,450				Quenched in iced brine from 900° C.
12	Initial size. 0.1717 diam.	Turned down after quenching to 0.112 inches diam.		72,400		18.	75.9	Quenched in iced brine from 900° C.
13	.066 x .066	Not.		114,000		16 per cent. in 3 inch.	72.6	Quenched in iced brine at minus 14½° C from 900° C.

These results are given in full in the *Engineering and Mining Journal*, vol. lxii., December 12, 1896, page 557, and vol. lxiii., January 30, 1897, page 111. The great tenacity of No. 13 is so surprising that, until it is corroborated by further tests, we must be cautious in accepting it.

The tensile tests were made for me by a trustworthy public testing company. The tenacity of No. 13 was originally reported to me as 114,000 pounds per square inch, and I published this result. But on repeating the experiments since these remarks were first printed, the highest tenacity which I find in this hardened wire is 66,154 pounds per square inch in a fragment of No. 13, 63,476 in

their skin, each of which should lessen stress, the effects of the sudden cooling are, if anything, intensified. Indeed, in two cases it has been found that a cylinder cut from the axis of a hardened steel bar is stronger—in one case enormously stronger—than the bar as a whole, though of course much less quickly cooled.*

Here, then, we have strong suggestions that the allotropic change indicated in A has been in part restrained by the very sudden cooling, with the consequence that our suddenly cooled metal retains part of the high-temperature, strong, brittle β iron.

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another wire, and 91,800 in a fragment of a wire which had already been tested. On investigating the matter I find that the tenacity of 114,000 pounds per square inch originally reported to me is wrong, and has been obtained by the testing company through an error in calculation.

In several cases I have failed completely to harden this wire, probably because it cooled unduly after leaving the heating furnace and before reaching the quenching bath.

* It may be urged that while stress may cause this strengthening and embrittling, yet for each set of conditions there is a certain definite intensity of stress that will produce the greatest strengthening effect, and that in these bars the stress may have exceeded this definite quantity. Further, that in reducing this stress by turning down the outside or boring out the center, I simply lowered the stress toward that specific amount which would give the greatest strength, and hence that the fact, that I increased the strength by reducing the stress, is no proof that the stress itself is not a cause of this strengthening. In view of this, I do not regard the fact that this reduction of stress which I caused actually increased the tenacity of the bar as a whole, as at all conclusive evidence that the strengthening effect of the sudden cooling is not due to stress: but I regard it as important contributory evidence. My main reliance is on the fact that, so far as we know, no other metal undergoes comparable strengthening when suddenly cooled, although they ought to undergo comparable stress. But I must admit that we cannot rely strongly on this, because we have so little evidence as to the influence of sudden cooling on the other metals. A few preliminary experiments which I have made with copper and platinum, while too meager to afford strong evidence, certainly suggested that these two metals were affected by sudden cooling to a much higher degree than is generally believed.

The experiments to which I have just referred as showing that boring out the interior and turning down the skin of suddenly-cooled nearly-carbonless iron did not lessen the tenacity, are represented in the table in the last preceding footnote.

In making a like experiment with a bar of steel containing 0.39 per cent. of carbon, I obtained the following results:

	Tensile Strength, Pounds per sq. in.	Elongation per cent.	In.
The annealed bar, 0.734″ diameter, .	92,900	31.2	2
The hardened bar, 0.74″ diameter, .	118,000	0.8	2
Core of the hardened bar, 0.565″ diameter,	141,000	0.0	2
Core of the hardened bar, 0.259″ diameter,	248,000	3.0	0.5
Segments of the exterior shell of the hardened bar, . . . 1st experiment,	165,000	.....	.....
Segments of the exterior shell of the hardened bar, . . . 2d experiment,	167,000	.....	.....

(*The Metallurgy of Steel*, Howe, 1890, pages 32 and 33).

There appears to be a disposition to break the force of evidence in this line by referring the strengthening, which sudden cooling causes, to any minute quantity of any element present, be it manganese, silicon or any other. One friend, more ingenious than logical, would attribute the strengthening of this 0.022 per cent. carbon-steel of mine to the 0.10 per cent. of copper which it contains. Let us look at the general features of the situation.

The properties—not only of iron, but of the other metals—are profoundly affected by the presence of impurities. These impurities, however, do not, so far as we know, confer the hardening-power, or any semblance of it, on any metal except iron. (With iron I include nickel and cobalt, the metals which, like iron, lose their magnetic properties—*i.e.*, become allotropic—at high temperatures.)

The hardening-power thus characteristic of iron is due to the retention of a high-temperature condition. That condition is either a peculiar high-temperature compound of iron with other elements, or a high-temperature condition of the iron itself; *i.e.*, allotropic iron.

Iron alone gives evidence of allotropy at high temperatures.

If carbon alone conferred the hardening-power on iron, we could reasonably say that the high-temperature condition preserved by sudden cooling was a special high-temperature carbide. But if we find that many different elements (carbon,

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I originally attributed these latter results in large part to stress. Further reflection, however, has convinced me that I overrated the influence of stress; and, especially in view of Moissan's discovery that carbon can be retained in an allotropic form of diamond by quenching highly carburetted iron, I now incline to the opinion that the greater strength of the central core is due to the retention of the iron in its allotropic condition, by the enormous pressure in the center of the bar. The center of the bar must, of course, cool much less rapidly than the outside, and common experience shows us that it is very much softer than the outside. These experiments of mine, however, indicate that the interior is very much stronger than the outside. I explain these facts by supposing that the rapidity of cooling gives a larger proportion of hardening carbon in the outside than in the interior, which agrees with our observation of the well-known intense hardness caused by hardening high-carbon steel. Further, that the intense pressure in the interior outweighs the slower cooling there as regards preservation of the allotropic iron; and this agrees with the observations that this allotropic iron, as in the case of manganese steel, while not particularly hard, is very strong. I am now pushing these experiments farther.

I must admit, however, that this whole line of evidence is as yet weak. We need much fuller information before we can put much reliance on it.

manganese, copper, etc.) all give the hardening-power, it is probable that they all act by preserving the allotropic condition of iron. For we can see no reason other than allotropy why many different elements should happen to form with iron special strong high-temperature compounds, when no element forms such compounds with any metal other than iron, or confers the hardening-power on any other such metal. Allotropy, however—shown to occur in the case of pure iron, by the loss of its magnetic properties and by its retardations in cooling—is made a more reasonable and probable explanation of the hardening-power, the greater the number of elements which are independently found capable of conferring this hardening-power. For while it is unnatural that all should happen to form strong, hard, brittle, high-temperature compounds with iron, unless allotropy of the iron itself be the cause of those compounds, it is, on the other hand, reasonable and natural to suppose that if one—carbon—tends to preserve this known allotropic state of iron, others should have a like catalytic effect.

Thus, the fact that manganese, like carbon, increases the hardening-power of steel, argues, in this view, for the allotropic theory, harmonizing on that theory perfectly with the fact that 12 per cent. of manganese, as in manganese-steel, both prevents the retardations of cooling (the signal of the change from the high-temperature non-magnetic to the low-temperature magnetic state) and preserves the unmagnetizableness so characteristic of the high-temperature state.

Thus, whichever way we look at it, the strengthening of these low-carbon irons argues for allotropy; though this line of reasoning needs much fuller evidence to make it really cogent.

C. On (1) heating a series of like steel bars to a given temperature above the upper retardations, (2) cooling each of them slowly under like conditions, (3) arresting this slow cooling at various temperatures by sudden cooling, and (4) then examining the properties of the suddenly cooled bars, it was found that, while much of the hardening power was lost at the lower retardation, Ar_1, simultaneously with a marked change in the condition of the carbon, yet a considerable loss of the hardening power occurred at the upper retardations, without

change in the condition of carbon.* Now, though the carbon theory readily explained why part of the hardening-power was lost at Ar_1, it certainly failed completely to explain why another part of it was lost at the upper retardations without change in carbon-condition. The allotropic theory, however, explains this latter fact readily; for it is at these upper retardations that the allotropic change appears to occur; and hence, if allotropy be a cause of hardening, we should expect part of the hardening-power to be lost here.

D. The magnetic phenomena have been shown, by a most beautiful piece of reasoning, to accord with the allotropic theory.†

First, we must distinguish clearly between the temporary and the permanent magnetism. The former decreases as the carbon-content of the metal increases, and is greatest in pure iron; the latter, occurring chiefly in hardened steel, increases at first with the carbon-content.

The normal low-temperature α state of iron is magnetizable; the allotropic high-temperature β state is unmagnetizable.

Temporary magnetism is due, in this view, to polarizing the particles of the soft α iron, to swinging them into a definite position relatively to the magnetizing force, just as pieces of loose iron-filings are thus swung. Permanent magnetism is due to retaining these particles of α iron in this specific position, and this, in turn, is due to the intermixture of the unmagnetizable particles of β iron with the magnetizable ones of α iron. The inert β iron, unmoved by the magnet, acts as a viscid matrix in which the particles of α iron are encased; and it becomes the more rigid the more there is of it. It thus impedes the magnetization of the α iron; but once this has been magnetized, and its particles have thus been swung into a given direction, the matrix of β iron restrains these particles thus polarized from swinging back into their original position, and so makes them permanently magnetic.

Let us now see how the more important facts taken separately agree with this conception.

The presence of manganese lowers the upper retardation-

* The writer, *Jour. Iron and Steel Inst.*, 1895, No. II., pp. 291 *et seq.* Prof. J. O. Arnold has confirmed this, though apparently unconsciously. (*Idem*, 1896, No. I., p. 203.)

† Osmond, *Phil. Mag.*, 5th series, vol. xxix., p. 511, June, 1890.

points, which as we have seen are identified with the change from the non-magnetic β iron to the magnetic α iron, and if enough manganese be present, as in Hadfield's manganese-steel, these points are brought so low that this change from β to α is so slight as to escape detection; it is nearly suppressed. The rôle of manganese thus is like that which the allotropists assign to carbon, that of a brake, restraining catalytically the change from the high-temperature β state to the normal α low-temperature state.

It ought, therefore, to be very difficult to magnetize manganese-steel, and it should even then be but slightly magnetizable, because the little α iron present is so rigidly held by the great mass of β iron. And this is the case. Manganese-steel can be magnetized appreciably only by enormous magnetizing force, and even then the magnetization is but slight.

Before going further, let us note a fundamental part of the allotropic theory, that carbon hinders the non-magnetic β iron from changing back to α even during slow cooling, and much more effectively during sudden cooling.

This explains readily why the temporary magnetism decreases progressively as the carbon-content increases; because the more carbon is present, the more β iron remains even after slow cooling, and by difference the less α iron there is to be temporarily polarized.

It also explains readily why hardened steel has so much less temporary magnetism, but so much more permanent magnetism, than the same steel unhardened. For the sudden cooling which induces the hardening, further greatly hinders the change from β to α iron, thus giving us an abundant matrix of β iron; but, in spite of the sudden cooling, a large quantity of iron changes back to α, so that we have just the conditions which permit permanent magnetism—the joint presence of polarizable α iron, and of a viscid matrix of β iron to hold the α iron in its polarized position, after we cease to apply the external magnetizing force.

In the same way we understand why the permanent magnetism of hardened steel increases with the carbon-content up to a certain point. For as the carbon increases, so will the matrix of β iron increase, and the more rigidly will it, therefore, hold the polarized α iron in its polarized position.

But, of course, when the increase in the quantity of β iron goes beyond a certain point, then its effect in increasing the firmness with which the polarized particles of α are held fast, and in thus increasing the permanent magnetism, becomes outweighed by the corresponding diminution in the quantity of α iron present to be held polarized. Further increase in the carbon-content should therefore decrease the permanent magnetism of the suddenly cooled metal, and this appears to be the case.

In short, this theory indicates that, as the carbon and manganese increase, the permanent magnetism of the suddenly-cooled metal, starting from zero, should first increase, then reach a maximum, from which it should again descend to zero; and this remarkable theoretical inference appears to accord with our observations.

To recapitulate, we have four things:

1. The retardations in perfectly carbonless iron;
2. The hardening-power of almost carbonless iron;
3. The loss of hardening-power at the upper retardations without corresponding change in carbon-condition, and
4. The magnetic phenomena;—

all of which seem in complete harmony with the allotropic theory. But on the carbon-theory the first is wholly inexplicable; the second and third are explicable only with difficulty; while, as to the fourth, no explanation on this theory has yet been offered, so far as I know.

It appears to me thus that allotropy stands to-day in the position of natural selection, as so completely and readily explaining well-verified facts, which thus far are not explicable readily, if at all, on competing theories, that we are compelled to recognize it as a probable cause. But in neither case are other contributory causes excluded by any of our evidence. To-day very many cautious but advanced thinkers, while recognizing natural selection as a powerful cause in evolution, are by no means satisfied that it is the sole cause. And this position, which appears to me wholly logical, I hold with regard to the allotropic theory.

Two chief considerations have led me to this view, that the condition of carbon also probably plays a very important part in hardening.

1. Even in case of steels which undergo the upper retardations in addition to Ar_1, much of the hardening-power survives these upper retardations, which mark the allotropic change, and is lost at the time when the carbon changes from the hardening to the cement or non-hardening state, at the retardation Ar_1, which has already and independently been well identified with that change in carbon-condition.

At first this seemed very cogent; but on reflection I must confess that it loses much of its force. The allotropists may argue, and with reason, that though in cooling much of the iron changes from β to α at the upper retardations, yet much, and in some cases most, of it may remain β till Ar_1 is reached, and there change back to α with the change in carbon-condition. They may further point to the fact that the loss of the hardening-power does not agree closely, even at Ar_1, with the change in carbon-condition, but rather lags behind it, indicating that the carbon-condition is not the primary cause, but only a concomitant, or at best a proximate cause of the loss even of this part of the hardening power. They may further point out that the distribution of the hardening carbon in 0.20-per-cent. carbon-steel, when quenched between the upper retardations and Ar_1, does not satisfactorily explain why the metal is so much stronger than when quenched below Ar_1. For this carbon appears to be distributed in the form of isolated specks of carbide, and it is difficult to understand how these unconnected specks can so greatly increase the tensile strength of the metal.* Specks of strong steel suspended in a bar of wax should not greatly increase the tensile strength of that bar as a whole. The allotropic theory, however, explains this with ease, by simply assuming that part of the change from β to α occurs at and even below Ar_1,† thus explaining why the loss of the hardening-power may lag behind the carbon-change.

They may further argue that, after all, we have no positive demonstration that the apparent change in carbon-condition is a real one; for it rests on a difference in the behavior of the carbon towards chemical reagents, and this difference may be due solely to the known difference in the mechanical mode of

* A. Sauveur and the writer, *Jour. Iron and Steel Inst.*, 1896, No. I:, p. 176.

† I appreciate that such assumptions must be pretty exasperating to those who, with a persistency which would be commendable were a moral principle at stake,

distribution of the carbide, and this in turn may be governed by the proportions of β and α iron present.

This piece of evidence thus appears rather equivocal. I do not see that it really opposes either theory strongly.

2. The other of the two chief considerations to which I have just referred is this. The allotropic theory explains the permanent magnetism of hardened high-carbon steel, and the unmagnetizableness of manganese-steel, by holding that the former contains both much of the magnetizable α iron, and also much viscid β iron, while the latter contains hardly any α, and therefore consists almost solely of β iron. Now, if this were true, and if β iron were the sole hardening agent in hardened steel, then manganese-steel ought to be much harder and much more brittle than hardened steel; while in fact it is much softer and incomparably less brittle. The carbon-theory may meet this case fairly, by holding that hardened carbon-steel is hard and brittle because the non-manganiferous martensite, or what Professor Arnold calls sub-carbide, is intensely hard and brittle, but that a manganiferous martensite, or an iron-manganese sub-carbide, is not.

It is, of course, a serious thing for the allotropists that the very compound which should contain the most β iron lacks the two most characteristic qualities of hardened normal high-carbon steel, its glass-hardness and brittleness, with which we are all so familiar.

The allotropists may indeed make a not unreasonable reply, somewhat thus: "Of course the presence of some 14 per cent. of manganese, which is about what this unmagnetizable manganese steel contains, whether it enter into the molecule of β iron, or simply accompany it in solution or intimate mixture,

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refuse to recognize allotropy even as a possible contributory cause of hardening. But they must remember that, just as the motion and properties of an invisible Neptune could be learned only through the perturbations which it caused in the motion of other planets, so the behavior, properties and influence of an undetectable allotropic β iron cannot be formulated *a priori*, and can be learned only through the perturbations which it causes, *i.e.*, through the discrepancies which it causes between the degree of hardening and the condition of carbon.

The allotropists, on the other hand, should recognize that this very inability to check their speculations by direct determination, either qualitative or quantitative, of the actual allotropic condition of the iron, demands of them great caution in thus speculating, and demands of their necessarily circumstantial evidence that it compensate by its completeness for its indirectness.

naturally should have a great effect; and there is no reason why it should not have this specific effect of lessening the hardness and brittleness. Hence in spite of its being less hard and brittle than hardened high-carbon steel, manganese-steel may yet have much more β iron."

"Moreover, the carbide of iron, cementite, which we clearly recognize as a substance much harder than hardened steel, contributes to the hardness of carbon-steel, both when hardened and when unhardened. It may, indeed, contribute more powerfully when in hardened steel, because there it is more uniformly distributed than in the same steel unhardened. It is but natural that, if manganese enter into this carbide, it should lessen its hardness."

But this explanation, while neither illogical nor unreasonable, seems to me labored and not cogent.

Therefore, while I think it probable that the strengthening of practically carbonless iron is due chiefly to allotropy, and that the hardening of the low-carbon steels in general is in considerable, and often in large part due to allotropy; and while I hence infer that allotropy probably plays an important part in the hardening of the high-carbon steels also; and while it seems much the most reasonable explanation of the magnetic phenomena: yet the enormous increase in hardness which we get on suddenly cooling high-carbon steel, and nowhere else, roughly proportional as it is to the carbon-content, seems to me, especially in view of the relative softness of manganese-steel, to raise a presumption, which our present evidence and argument fail to overcome, in favor of the view that a special high-temperature combination of iron and carbon contributes powerfully to the hardening of carbon-steel in general, and especially is a very important cause of the two most striking effects of hardening, glass-hardness and brittleness.

But, with our present imperfect evidence, such views should be provisional. While I cannot now see how it is possible that the condition of carbon ever should explain all the facts, it seems to me by no means improbable that further evidence may show that allotropy is the sole primary cause of hardening itself, *i.e.*, of the *change** induced by sudden cooling.

* I say of this *change*, because it seems in the highest degree improbable that it can ever be shown that the greater hardness of high- than of low-carbon steel, when annealed, is not in large part due to the presence of iron-carbide, cementite.

Be it clearly understood that these two possible causes are certainly not mutually exclusive. Those who admit that one or the other may be the cause, must admit that both may contribute. Holding this to be the most probable explanation, I think that our aim now should be to learn in what proportion each does contribute under given conditions.

I do not see why we are forced to choose one and reject the other any more than, because I cannot question the power of natural selection, I must hold that no other force contributes in evolution.

Some, among them valued friends, have represented me as trying to reconcile theories and even parties. The compliment is left-handed and, I think, gratuitous. The inquirer has to reconcile facts, not factions. In considering scientific phenomena, he should reject the motive of reconciling theories or people. His mental processes should be governed by one motive only, the search for truth. With given data, this sole motive must lead a given mind in one certain path. That any other motive influences that mind, by definition implies that it deflects that mind from that path and dishonors it.

The carbo-allotropic theory, as I first formulated it, suggested the existence of a carbide of an allotropic state of iron. The objection which was raised, that the retention of an allotropic modification in combination is opposed to chemical laws, seems to me neither tenable, nor, indeed, intelligible. When challenged, the gentleman who raised it has failed to cite a single law thus opposed. In fact, our present conceptions are in part based on the retention of isomerism in combination, and this differs from the retention of allotropy only in relating to specific groupings of *unlike* atoms instead of to specific groupings of *like* atoms. But the retention of specific groupings of like atoms, far from being opposed to our theories, in reality forms an important part of their foundation. For our views of the constitution of organic compounds really rest on the belief that the atoms of carbon in such hydrocarbons as benzene, C_6H_6, olefiant gas, C_2H_4, and marsh gas, CH_4, are grouped together in certain characteristic modes, and that these modes of grouping of these like carbon atoms persist in the derivatives of these hydrocarbons.

Further, not only has the idea that allotropy can be preserved

in combination long been a familiar one, but such chemists as Berzelius, Brodie, and apparently Berthelot, not to mention De Cizancourt and D. Berthelot, have asserted specific instances of such preservatiou. Indeed, Brodie's belief that carbon retains a peculiar allotropic condition in graphitic acid and the graphitates, is widely accepted as a reasonable explanation of the phenomena.

We may differ as to whether, in these specific cases, the preservation of allotropy in combination has actually been proved; indeed, few will now think Berzelius' evidence sufficient; but we cannot say that the conception is opposed either to chemical laws or to chemical theories.*

However, to this feature of the theory I attach little weight. It was put forth tentatively and with reservation, in the hope that it might aid in clearing up this complex problem. The feature to which I do attach importance is that both carbon-condition and allotropy play an important part in hardening.

Mr. Sauveur's explanation, put forth in the last two pages of his paper so ably, with such modesty, and with the caution so proper in a man of science, is most suggestive and instructive. I have gladly borrowed his ideas in what I have just said.

As I understand, he regards the variations in structure as the proximate cause, not only of hardening, but of the various changes induced in the properties of steel by thermal treatment. In this way important laws may be formulated, and for many this explanation of the cause of hardening may suffice.

I think, however, that Mr. Sauveur and some others will wish to peep behind this curtain and ask what the primary causes of these structural changes are, and I understand Mr. Sauveur's words, "the saturation-point of carbon for iron is lowered at the two upper critical points," to admit that these structural changes are, after all, only proximate causes, and that behind them must lie at least one primary cause, residing in the relations between iron and carbon. These words of his seem to imply that, at these critical points, either (1) some carbide of iron or (2) a solution or a mechanical mixture, is dissociated; or (3) that both a carbide and a solution or mechanical mixture are so dissociated. If, now, we push behind these proximate

* Those interested in this question may well consult D. Berthelot's essay, "De l'Allotropie des Corps Simples," Steinheil, Paris, 1894. Full comments of mine on it may be found in the *Jour. of the Iron and Steel Inst.*, 1897, No. I., pp. 212-223.

structural causes of hardening to seek the primary cause, the first of these two suppositions would, I fancy, land us with the carbonists, the second with the allotropists, and the last with the carbo-allotropists.

The reason why the second would land us with the allotropists is that, as the temperature rises, the dissociation of a solution or mechanical mixture should occur, not with a bound, but progressively, unless some component changes its state or changes allotropically. As no change of state occurs, the suddenness of the changes at the upper retardations thus implies that, if it be not a carbide that dissociates, then either the iron or the carbon changes allotropically. But, while we have no evidence that carbon here changes allotropically, (1) the retardation of carbonless iron accompanying the loss and recovery of the magnetic properties during heating and cooling respectively, and (2) the great strengthening of almost carbonless iron by sudden cooling, are strong evidence that the iron does change here allotropically.

Prof. Arnold's Theory.—I agree with Mr. Sauveur, both that Professor Arnold's experimental results are of very great value, and that they do not at all justify the inferences which he draws from them. The important novelty in his theory is its assigning to the carbide of iron, previously supposed to form at high temperatures, a distinctive name, subcarbide, and a special composition, $Fe_{24}C$. This formula is based chiefly on the fact that, as the carbon-content rises, some important properties of steel reach a maximum in the neighborhood of 0.90 per cent. of carbon, corresponding nearly to the composition $Fe_{24}C$., and that the pearlyte of unhardened steel is approximately of this composition, and is somewhat segregated, instead of being uniformly distributed.

The existence of this maximum-point, as regards tenacity, was previously known; but Professor Arnold's invaluable evidence showed not only that it applied to some other properties, but why it applied to them. Thus far he deserves great gratitude.

But we cannot justly infer from the existence of this maximum-point, which he happily calls the saturation-point, that the subcarbide has this definite formula. The martensite, which contains this subcarbide, varies very widely in composition. This wide and continuous variation shows that martensite is

either a solution or a mechanical mixture. Whichever of these it be, the existence of a maximum-point corresponding to 0.90 per cent. of carbon is no more evidence that one component of that solution or mixture also has that composition than the existence, in a series of granites, of a maximum compressive strength for a certain given ultimate composition would be evidence that one component of granite always had that composition.

I fail to see that the composition and the partial segregation of the pearlyte have any bearing whatever on the question.

We must recollect that probably but few of those who accepted Professor Arnold's theory were thoroughly familiar with the evidence and arguments supporting the allotropic theory. The language of some of his best supporters shows that they were not. Indeed, much of the most cogent evidence has come to light since Professor Arnold wrote.

Nomenclature.—If reason additional to those given by Mr. Sauveur for the use of the names ferrite, cementite and pearlyte be needed, it exists in the fact that these names were proposed with the written approval of Dr. Sorby, the discoverer of these substances. Therefore, according to the well-recognized law of priority, these are the names *de jure*. I believe that they are also the names *de facto*, *i.e.*, that they are now used by a majority of the best writers. If this be true, the use of other terms is a hardly justifiable impediment to the advance of knowledge.

The objection to the name martensite, that this substance is not of uniform composition, but varies within wide limits, I cannot comprehend. Granite, syenite, trachyte, and dozens of other lithological species, have not definite but widely varying compositions. The point to remember is that martensite is a distinct entity, be it of constant or variable composition; and that it is of such importance that the convenience of language demands that it shall have some distinctive specific name. To suppose that specific names are to be given to things only of constant composition is to ignore lithology completely. There is no reason why lithological entities, as well as mineralogical entities, should not in metallography receive distinct specific names.

MR. SAUVEUR: I am gratified at the amount of valuable dis-

cussion called forth by my paper, and grateful for the friendly and indulgent tone of the criticisms.

From a careful reading of these remarks it now appears that while there are four main theories put forward to explain the hardening of steel by sudden cooling from a high temperature, yet when we come to examine the question closely we find that there are actually seven different ways of accounting for the phenomenon. These are summarized in the following table, in which the main theories are called respectively the carbon theory, the carbide theory (this includes Prof. Arnold's), the allotropic theory, and the carbo-allotropic theory. The names of the metallurgists supporting in the present discussion the various explanations are given in the last column.

	Condition of Carbon and Iron at a High Temperature, Which Being Retained by Sudden Cooling Produces Hardness.	Supported by.
I. Carbon theory	1. Carbon in its normal state (graphitic?) dissolved (decomposed into its ions?) in iron.	Ledebur.
	2. Carbon in a hard state (diamond?) diffused (dissolved?) in the iron.	Scott (tentatively).
II. Carbide theory	3. Carbide $Fe_{24}C$ forming totality of the mass in saturated steel (0.90 per cent. of carbon) but diffused through iron in softer steel.	Arnold.
	4. Carbide Fe_3C diffused through iron.	Sauveur (tentatively).
III. Allotropic theory	5. Carbon diffused (dissolved?) through a hard allotropic condition of iron.	Osmond and Roberts-Austen.
	6. Carbide Fe_3C diffused through hard allotropic condition of iron.	
IV. Carbo-allotropic theory	7. Carbide of allotropic iron.	Howe (tentatively).

It is believed that by summing up in this way, at the begin-

ning, the views expressed in this discussion, the following remarks will gain in clearness and interest.

Prof. Ledebur takes exception to my way of stating the carbon theory, yet I do not see anything in his presentation of the question which is not included in mine, and especially in Mr. Howe's description, which I quote on page 890. I concede that, with reference to the condition of the carbon above the critical range, "hardening state" expresses Prof. Ledebur's view more accurately than "hard state" (which, indeed, was a slip of the pen), although "dissolved state" would be better than either.

And here, at the outset, we very much feel the need of more positive statements with regard to what the school of metallurgists to which Prof. Ledebur belongs means by a "solution" of iron and carbon. Do they understand by it a mere diffusion, although extremely minute, of the carbon in a matrix of iron, or do they consider the carbon as being actually dissolved into the iron, in the chemical sense of the word; the carbon in such case in its normal state (graphitic?) being decomposed into its ions (graphitic ions?) if we accept the electrolytic theory of solution? That this question must be asked and that it can probably receive only a confused and unsatisfactory answer is a further illustration of our ignorance regarding the true nature of solution.

From Prof. Ledebur's remarks we certainly gather the impression that he attributes the hardening of steel not so much to a hard condition of the carbon as to the fact that the carbon is dissolved in the iron. Might it not be argued with nearly as much reason that the saline taste of sea-water (liquid or solid) is due not so much to the taste of the salt (indeed, is quite independent of that taste) as to the fact that the salt is dissolved in the water? We know, however, that the salted taste of the solution comes from the taste of the salt; that if the salt did not have a saline taste no amount of dissolution could impart such taste to the solution, and that the more "salted" the salt the more pronounced will be the same taste in the solution. Must not the attributes of a solution be intermediate between those of its components? and do not the same laws apply to the mechanical properties of what might be termed "solid or frozen solutions?" Can the mere solution of two substances, neither of them very hard, produce a whole of ex-

treme hardness? Is there any known fact which would justify us in answering the question in the affirmative? Would it not be strongly suggestive, even conclusive, of chemical combination?

If it be true, then, that mere solution of two relatively soft substances cannot produce a compound of extreme hardness, we must assume that one of them is itself in a very hard state in order to explain the hardness of the product. The allotropists say this one is the iron; the carbonists must say that it is the carbon. If the dissolved carbon were of the soft graphitic variety, no amount of dissolution could produce the extreme hardness of the resulting solution.

In a word, while it might be admitted that carbon in a dissolved condition would increase the mineralogical hardness of the mass more than when disseminated throughout the iron in relatively large segregated particles, nevertheless it is the hardness of the carbon itself which must be the principal cause of the hardness of the mass. Therefore, we place the "carbonists" in the most favorable position to defend their theory by assuming that the carbon of quenched steel is of adamantine hardness. Even then, however, it is difficult to understand, and Prof. Ledebur's remarks do not in any way help us in this direction, how one part of carbon, however hard, dissolved throughout 200 parts of soft iron (C 0.50 to Fe 99.50 per cent.), can produce the intense hardness of steel of such composition, which has been suddenly quenched from a high temperature. Until this point has been brought within the range of our comprehension, the carbon theory must still fail to satisfy many metallurgists.

Will it turn out that, after all, the carbonists mean that the carbon in hardened steel is combined with a certain amount of the iron, forming a definite carbide, itself very hard, and dissolved through the iron? This would make their hypothesis identical with what is here called the carbide theory. Prof. Ledebur's description scarcely warrants such an inference, and if it be, nevertheless, correct, it should be stated positively and without ambiguity.

Professor Ledebur's account of the separation of the dissolved carbon from the general mass during slow cooling is not absolutely correct. Supposing that the carbon is in solution as such when iron is in the molten state, the microscope shows

that on solidifying the metal can only retain about 0.90 per cent. of carbon in the dissolved condition (martensite). On slow cooling, during the critical range, the martensite is changed into pearlyte (hardening carbon into cement carbon), yielding, besides, an amount of free carbonless iron (ferrite), which is the greater the less carbon there is in the martensite. If the steel contain more than 0.90 per cent. of carbon, the excess on solidifying separates as Fe_3C (cementite). When a large excess of carbon is present, the cementite segregates in irregular masses distributed through a matrix of martensite in quenched steel, and of pearlyte in slowly cooled steel (Figs. 15 and 16, Plate I.). When the steel contains only a small amount of carbon above 0.90 per cent., then, and then only, does the free cementite form a net-work (a membrane) around the grains of pearlyte (Fig. 14, Plate I.).

Professor Ledebur says that the carbon theory rests upon the investigations of Rinman, Åkerman, Müller, Abel, Osmond and others. By this he can only mean that the theory rests upon the fact that the behavior of the carbon of hardened steel is different from that of the carbon of unhardened steel, when treated with cold dilute acids, from which it is inferred that the carbon in hardened steel exists in a peculiar condition (hardening state). This is a fact which the above-named scientists have conclusively established, and which Professor Ledebur has confirmed by numerous analyses; but I am not aware that they have done anything else to prove the correctness of the carbon theory.

Professor Ledebur says that the microscopist must be "able to segregate and examine separately the bodies which he believes himself to have detected" before much weight can be accorded to his methods. If the same threat were held over all the sciences in which the microscope plays an important part, I fear their progress would be seriously impeded. But the criticism is not so very weighty in this case, after all; for the microscopist simply claims to find in steel four principal micro-constituents, namely, ferrite, cementite, pearlyte and martensite. The first, he says, is carbonless iron; the second, the carbide Fe_3C; the third, an intimate mixture of the first two; while the nature of the last constituent is still a subject of controversy.

With regard to ferrite, it would be superfluous to offer here any evidence to prove that it is iron free from carbon. It can be found segregated in any quantity, since it forms nearly the whole mass of wrought-iron; neither shall I attempt to again recall the numerous evidences which go to show that the hard constituent which the microscopists distinguish in the structure of steel, and which they call cementite, is the carbide Fe_3C. They have often been stated, and are conclusive. Moreover, by dissolving very slowly a thin plate of hard steel the cementite will be left undissolved in the same position which it occupies in the structure, and if analyzed will be found to be the carbide Fe_3C. This meets Professor Ledebur's requirement.

It is, perhaps, not so easy a task to convince the reader that the constituent pearlyte is merely an intimate mixture of ferrite and cementite; yet the evidences offered are quite conclusive, and the only one which I shall give here ought, it seems to me, to remove all remaining doubt. If a piece of saturated steel be annealed, it is composed entirely of lamellar pearlyte, made up, we know, of very thin layers, alternately hard and soft. All indications lead us to the conclusion that the hard layers are cementite and the soft ones ferrite. But if, like Professor Ledebur, the reader refuses to believe that such is the composition of the thin layers, unless he can segregate and analyze them, let him dissolve this steel in cold dilute hydrochloric acid. He will obtain a residue containing practically the totality of the carbon, and made up of thin plates, which, upon being analyzed, he will find to have a composition corresponding to the formula Fe_3C. Does not this prove that the hard layers of pearlyte are cementite, and that the soft ones, therefore, are iron, or ferrite?

The systematic study of these four constituents, of their mode of occurrence, of their distribution and relative proportions in steels of different composition and submitted to various treatments, has thrown much light upon questions imperfectly understood, and seems destined to clear away many of the difficulties which still surround the physics of steel. While it is difficult at the present time to form a correct estimate of the practical value of metallography, those metallurgists who have kept in touch with its developments will admit, I think, that it will probably stand some day side by side with chemical analy-

sis in importance and usefulness. "Of this ultimate result," says Professor Martens, "no one can doubt, who himself has carried on these researches, or even has had the opportunity to study a collection of metal slides." The two methods of testing will complement each other, for it is precisely where chemistry fails to give us the needed information that the microscope comes to our assistance.

I think that Mr. Hadfield is not what has been called here a "carbonist," but rather supports some form of the carbide theory, for he does not attribute the hardening of steel to the action of carbon as such, but to the influence of some carbide of iron, itself intensively hard. With regard to Mr. Hadfield's reference to a note written by Mr. Howe and myself,* I must say here that I shared without restrictions the views expressed in that communication, which views do not in any way conflict with those given in the present paper. Our concluding remarks were of a very conservative and undogmatic character, as will be seen from the following quotation:

"These suggestions are offered in the hope that they may aid in finding the true explanation of this complex matter. We will not dignify them with the name of hypothesis, much less with that of theory, believing that an unfortunate effect of insisting prematurely on theories is to bias the theorizer for, and almost everybody else against, his theory, and thus to interfere with the judicial frame of mind in which these questions should be approached.

"For the same reason we express no opinion here as to the merits of the allotropic theory, preferring to await further information, and especially because we are ourselves studying microscopically a matter which has lent this theory one of its very strongest supports, viz., the great strengthening which almost carbonless iron undergoes on quenching."

Mr. Scott calls attention to an apparent disagreement between Fig. 1, illustrating the position of the critical points, and Fig. 5, showing the changes of structure occurring during the slow cooling of a steel containing 0.22 per cent. of carbon. The changes of structure taking place in this steel correspond exactly with *its* critical points, and in every respect confirm my proposition that "each critical point is accompanied by a structural change which begins and ends with it, and that in the ranges of temperature where there is no critical point we find no structural change." In this special case, however, the

* "Further Notes on the Hardening of Steel." *Jour. Iron and Steel Institute*, No. 1, 1896, p. 170.

critical points do not occupy the positions they should have, according to Fig. 1. As I have stated, this last diagram was obtained by platting the results of previous investigators as well as my own; and it will be found that with a *very few exceptions* the evolutions of heat always occur within the limits here indicated. Mr. Howe's steel is apparently one of these exceptions. Ar_3 is here missing (that is why we do not find any structural change between 800° and 750° C.); and, as Mr. Scott remarks, Ar_2 and Ar_1 occur at temperatures lower than usual. This steel contained a relatively large amount of manganese (1.19 per cent.) and of other impurities (0.31 per cent. silicon), which probably accounts for the abnormal position of the critical points. Indeed, Professor Arnold, on repeating the experiments carried on by Mr. Howe with a steel containing the same amount of carbon but much less impurities (0.05. per cent. manganese and 0.05 per cent. silicon), found three distinct critical points, occurring respectively at 782°, 733° and 686° C., which positions are in agreement with the diagram of my Fig. 1.

Mr. Scott thinks it hardly possible to account for the hardening of steel by a mere diffusion of the carbide Fe_3C in the iron, and still he considers it reasonable to suppose that diamond distributed through the iron in like manner might explain the phenomena. Now, while it is far from being my intention to insist that the diffusion of the carbide Fe_3C can produce the extreme hardness of quenched steel, it seems to me that it is just fifteen times harder to conceive that it is due to carbon in an adamantine condition, because the carbide Fe_3C (6.67 C + 93.33 Fe) would constitute a proportion of the mass fifteen times greater.

With regard to the hardness of cementite, I know of no reason whatever for supposing that it varies. That cementite is the hardest of all the constituents is certain, because it is always the one which stands in relief when a piece of steel is polished on a soft support, even in the presence of martensite, as in the case of quenched high-carbon steel.

Dr. Müller* compares the hardness of cementite to that of feldspar or No. 6 of Mohs' scale. Professor Behrens, I believe, also attributes to it a similar degree of hardness.

* *Stahl und Eisen*, 1888, vol. viii., p. 292.

I believe Mr. Scott would find it very difficult to cut the steel of Fig. 16 with that of Fig. 11. The stout layers of cementite of the former sample will resist abrasion, being harder than the martensite which constitutes the whole of the latter. A steel like that represented in Fig. 14 will easily be cut by that of Fig. 11, because here the plates of free cementite are extremely thin, and being also very brittle, will break off as they come in contact with the moving tool of martensite.

Mr. Scott ably revives the diamond theory, and whatever may be said against it, it is a more acceptable explanation of the hardening of steel than that offered by those carbonists who claim that carbon need not be hard to produce extreme hardness; that mere *solution* can account for that result.

With the explanation offered by Mr. Scott for the absence of diamond dust in the residue from the solution of hardened steel in acids I am in sympathy; for it is indeed exactly in this way that I would account for the volatilization of the Fe_3C which I believe to exist above the critical range, and therefore in the quenched metal. According to Professor Jenkins, however, such chemical activity could not exist, unless the carbide Fe_3C were in a dissociated condition, *i.e.*, no longer a carbide. On the other hand, Mr. Osmond tells us that ". . . the minute state of division of the carbide Fe_3C explains sufficiently well the readiness with which it could be attacked by hydracids. . . ." This purely chemical question, therefore, is not settled, and deserves careful investigation.

I have no doubt that good etchings can be made with very dilute acids; and I only meant to state that in order to obtain by this method results as good as those yielded by the concentrated acid or iodine process, more skill was required, and that the attempt was accompanied by frequent failures. At least, such has been my experience.

Professor Arnold says that the present paper has removed any doubt which could have existed in his own mind regarding the wisdom of caution in adopting the nomenclature used to describe the micro-constituents of steel, but at the same time he declares that the names "ferrite" and "cementite" are admirable, thus unreservedly endorsing half of such nomenclature. In the interest of metallography I welcome his partial endorsement. Professor Arnold sanctions, moreover, the use of

the term pearlyte, objecting only to its being applied to both varieties of the constituent, namely, to granular and lamellar pearlyte. He proposes to call the first one "sorbite," and to retain the term pearlyte for the second variety. I believe this course should be opposed vigorously, because we are evidently here in presence of two varieties of the same substance, and to give them two different names is decidedly objectionable. One might as well claim that a substance susceptible of crystallizing in two systems, or of assuming two different structural arrangements with corresponding variation of its physical properties, should bear also two different names. In certain steels, the two varieties of pearlyte pass into each other so gradually that it would be impossible to tell where sorbite begins and pearlyte ends, or *vice versa*.

Professor Arnold's objection, that when the constituent is granular, lacking therefore in interference-colors, the name pearlyte loses its fitness, has more weight, but is after all of little moment. There is certainly great justice in giving the name of sorbite to one of the constituents of steel; but if it be applied to the composite constituent, then it should be to both varieties, one being called pearly sorbite, the other granular sorbite.

If it could be conclusively shown that martensite is iron containing finely divided Fe_3C, it should of course become a third variety of pearlyte, and should be named accordingly. I did not consider, however, that on a mere suggestion regarding the composition of martensite I would be justified in proposing to change its name.

Professor Arnold objects to giving any name at all to martensite, because its properties are not constant. His reasoning could be applied to steel with as much force, for of steel also it might be said that "it is very soft and flint-hard;" that "it readily flows under crushing stress, but is incapable of compression"—therefore it should not have any name. Martensite is, however, a very distinct constituent, and should be treated as such.

Prof. Arnold will notice that I do not appeal to priority in the case of martensite. Such appeal referred only to the first three constituents. Hardenite, as Prof. Arnold says, has the right of priority. I have not adopted it for two reasons. The

first is, that while it will do very well for English-speaking metallurgists, it is meaningless and objectionable to those of other nationalities. The second reason is that Mr. Osmond was the first to study and describe, to any extent, the character and mode of occurrence of this constituent, and had therefore some right about the choice of a name.

Prof. Arnold still believes that the evidences point strongly to the existence of his sub-carbide $Fe_{24}C$; but he no longer insists that the dark areas of quenched steel (martensite) are always made up of that sub-carbide. Such a position was, indeed, untenable. He now admits that his carbide, which forms the total mass when the steel contains about 0.90 per cent. of carbon, in unsaturated or low-carbon steels, is dissolved or diffused through the iron, forming with it a micro-mechanical mixture.

Prof. Arnold gives us his reasons for doubting the possible segregation into distinct masses (of iron and carbon) of the diffused carbon. He argues that if the carbon did exist evenly diffused through a matrix of iron, and if it did segregate into distinct masses, such arrangement being retained by sudden cooling, upon dissolving the metal we should find the carbon in the residue, while, on the contrary, it escapes as hydrocarbons.

What Prof. Arnold, in fact, here questions is the *existence* of diffused carbon, not its power of segregation; for it is not because it segregates that we should find it in the residue, but simply because of its presence in the metal. Had it not segregated previous to quenching, we should still expect to find it in the residue.

So Prof. Arnold has changed the issue; and although, of course, if it be proved that the carbon cannot exist, it follows that it cannot segregate, nevertheless, Prof. Arnold fails to derive from such reasoning any proof exclusively favoring his theory. He could at best make a case against the possible existence of free, undissolved carbon, but it would not, by any means, follow that the carbon and iron are combined so as to form the carbide $Fe_{24}C$. Moreover, as is made clear in Mr. Osmond's paper, the allotropists do not insist that the carbon is diffused *as such* through the *beta* or *alpha* iron; indeed, they are quite ready, not to say inclined, to believe that it exists in the form of a carbide.

As an evidence of the existence of the carbide $Fe_{24}C$, Prof. Arnold had offered the fact that when the steel contains about 0.90 per cent. of carbon it evolves more heat during the recalescence than does steel of any other degree of carburization. Upon this he had laid much stress. I thought to have shown, in the present paper, in a way that would satisfy any impartial reader, Prof. Arnold included, that such phenomenon has nothing whatever to do with the existence of the carbide $Fe_{24}C$. Prof. Arnold dismisses my arguments by saying that they are "merely opinions unsupported by any evidence."

I should be satisfied in letting the case rest here could I feel sure that all those interested in the controversy have studied the present paper carefully. It is quite probable, however, that many readers have gone over the ground hurriedly, and may, therefore, be led by Prof. Arnold's remarks into thinking that my conclusions were indeed "merely opinions unsupported by any evidence." Inasmuch as those conclusions, if they are sound, go to show that Prof. Arnold's theory is untenable, they have a general interest, and it is important then, for the sake of scientific truth, that the reader should not be misled.

Were it not for the above considerations, I should not feel justified in taking up the question again.

Let us examine again Figs. 9, 11 and 13 of Plate I., which show the structures of hardened steel containing respectively 0.35, 0.80 and 1.20 per cent. of carbon. The first steel is under-saturated, *i.e.* it is a mixture of martensite and ferrite; the second sample is saturated, *i.e.* it is made up entirely of martensite; the third is over-saturated, *i.e.* it is a mixture of martensite and cementite. Whatever the true nature of the phenomenon, we know that on slow cooling, during the critical point Ar_1, there is a decided evolution of heat, caused by the martensite being changed into pearlyte (see Figs. 10, 12 and 14). The ferrite of the under-saturated steel, as well as the cementite of the over-saturated sample, remains unchanged. *They do not contribute to the evolution of heat.* For the purpose of our argument, we may consider the ferrite and cementite as *thermally inert* constituents, the martensite alone being *thermally active.* The first and last samples, then, are mixtures of thermally inert and thermally active components, while the saturated metal is made up entirely of the active constituent. It

might therefore be easily anticipated that the saturated sample, which contains only the active constituent, will evolve more heat than either of the other two, which contain a certain amount of inert substance. Two pounds of a certain heat-evolving substance must evolve more heat than one pound of the same substance.

Prof. Arnold, however, considers this result startling, and offers it as conclusive evidence that the saturated metal is a carbide of iron, answering to the formula $Fe_{24}C$! But what bearing has this phenomenon upon the existence of such a carbide?

If there is a chemical change taking place at Ar_1, there is also a structural change; and Prof. Arnold's calling me to account for speaking of the heat evolved here as "heat of the structural change," appears trifling.

Prof. Arnold's answer to what he calls the "thermal portion" of my paper is disappointing. Where we had a right to expect arguments, he contents himself with a few uncomplimentary remarks concerning that part of my paper, and with a claim that he has been misquoted.

I contended: (1) that Prof. Arnold claimed that his sub-carbide $Fe_{24}C$ is formed during the upper retardation (Ar_3 or Ac_3), and (2) that it was not possible from his papers to decide whether it was at Ar_3 (during cooling) or at Ac_3 (during heating) that the phenomenon took place. As he now says that such ideas can only have originated in my imagination, I shall have to tax the reader's patience once more by again quoting from Prof. Arnold.

In the *Journal of the Iron and Steel Institute*, 1894, No. 1, on page 136, beginning at the 3d line, he says:

"The above remarkable facts are satisfactorily explained by the hypothesis that the point Ar_3 marks the formation of a sub-carbide of iron, whilst the point Ar_1 is due to the combination of the elements to form the normal carbide Fe_3C."

This is the quotation which Prof. Arnold wishes me to locate. On the same page he says:

"At first sight it seems that the author's tentative hypothesis, that the point Ar_3 is due to the chemical heat evolved by the formation of a sub-carbide, is at once decisively negatived by the alleged fact that electrolytic iron also develops the point in its most marked and individualized degree."

Then, as it is necessary for Prof. Arnold, in order to maintain his position, to show that the point Ar_3 does not exist in electrolytic iron, he enters into a long argument (page 136 *et seq.*), in which he claims that whenever the point has been developed there was a certain amount of carbon present, or else a relatively large amount of sulphur, which he says will produce an evolution of heat.

In a more recent paper (*Proceedings of the Institution of Civil Engineers*, vol. cxxiii., page 127, on "The Influence of Carbon on Iron") Prof. Arnold expresses himself as follows (page 154):

"... the heat evolved or absorbed at Ar_1 is due not to .... but to a carbonization of the sub-carbide $Fe_{24}C$ to Fe_3C or a reduction of Fe_3C to $Fe_{24}C$."

It is evident, in view of his previous remarks, that the point which Prof. Arnold has in mind here is $Ar_{3,2,1}$, not Ar_1. In other words, that the sub-carbide forms during the only retardation present in medium-hard or hard steel, in which case Ar_1 and Ar_3 are united. If he now insists that it is during the lower retardations Ar_1 that the sub-carbide comes into existence, it will only increase the confusion and illustrate more forcibly the speculative character of his theory.

Prof. Arnold asks me my method for distinguishing Ac_3 and Ar_3 in hard steel. I do not understand this request, as I cannot recall anything I have said which led him to believe that I could do it.

Concerning the crystallization-point of iron, Prof. Arnold claims to have been misquoted. The only ground for this claim will be found in a typographical error—the word "quenched" having been used instead of "finished." In some way the mistake escaped detection, but I felt confident that the reader would easily detect it, since the sentence, as printed, has no meaning whatever.

I have advanced the two following propositions:

1. On heating, the structure of steel becomes amorphous during the critical point. 2. On cooling from above the critical range it begins to crystallize as soon as the cooling begins.

In support of these two propositions I thought myself to have presented conclusive experimental evidence, but Prof. Arnold denies it, and the reader will have to judge for himself.

Prof. Arnold says that, according to my views, "iron heated,

say to 1200° C., will on cooling, say to 1100° C., consist of large crystals." On the contrary, I should expect, on account of the short period of crystallization, to find very small crystals.

Mr. Hibbard reminds us of the theory of Messrs. Langley and Metcalf, who attribute the hardening of steel to the existence of stresses in the suddenly-cooled metal. Hardened steel, Mr. Metcalf says,* "may be regarded as a congealed liquid in a state of tension; then all known phenomena could be accounted for, and all known conditions could be produced with certainty by well-known applications of heat and force."

It is quite purposely that the above theory was not considered in my paper. Whatever the true cause of hardening, it is certain that the hardening-power is gained and lost during the critical points. Whatever the nature of the changes which occur at the critical points, it is certain that the cause which produces the thermal disturbances in the metal is also the one which confers upon it its hardening-power. And any theory which not only fails to account for the retardations, but apparently takes no notice of them, cannot possibly hold its own in the present advanced stage of investigation.

A piece of steel quenched from a temperature just above the point of recalescence will harden to its full capacity, while, when suddenly cooled from a temperature just below that point, it does not harden. A difference of quenching-temperatures often less than 10° C.† may produce these enormously different results. Will it be urged that in a piece of steel heated to a high temperature and cooled, say to 675° C., before quenching, the "molecular motion" is so great that upon being "checked violently by sudden cooling" the cold metal is in such a state of tension as to account for its extreme hardness, but that if the steel be cooled a little more, say to 650° C., then the molecular motion has so much moderated that no increase of tension is imparted to the quenched metal, therefore no increase of hardness? The proposition is hardly conceivable. That hardened steel is in a state of internal stress seems, indeed, certain, but that this condition is the cause of the enormously altered properties of the metal is untenable. Again, as

* *Steel*, by William Metcalf, pp. 149 *et seq.*

† "The Heat Treatment of Steel," by H. M. Howe, *Trans.*, xxiii., 466.

has been pointed out, all metals at a high temperature are in a state of high molecular motion. Why, then, do they not harden on sudden cooling, in some degree comparable to the hardening of steel?

Finally, Mr. Howe now shows us that by removing those portions of hardened bars which should have received the greater amount of stress, not only the remaining portions were not softened, but in some cases they showed an actual increase of tenacity.

Mr. Metcalf contends, further, that the stress theory "is a good working-hypothesis for all use, and one which it is believed will always be the right one to follow, no matter what the final explanation of the remarkable phenomenon of hardening .... proves to be."

The test of a good working-hypothesis is, I take it, that if, in order to obtain a certain result, we place ourselves in the conditions demanded for it by the hypothesis, we succeed. Let us apply that test to the stress-theory.

This theory, as already stated, ascribes the hardening of steel to the fact that the cold metal is in a state of tension, such condition having been brought about by a sudden check of the high molecular motion of the metal at a high temperature.

It naturally follows, from such a hypothesis, that the higher the quenching-temperature the greater the stress in the suddenly cooled metal, and therefore the greater its hardness. Hence, in practice, in order to harden as much as possible, we should heat our steel to the highest possible temperature. But it is now well established that if the metal be heated sufficiently above the critical point to make it sure that the change occurring here has been completed, say to 800° C., further elevation of temperature will produce very little increase of hardness, *if any*.

The stress-theory should also lead us to expect that the hardening produced in the metal by sudden cooling would decrease *very gradually* with the fall of the quenching-temperature, down to the atmospheric temperature; whereas, the hardening-power disappears *abruptly* during the period of recalescence.

Again, a piece of steel quenched just above the critical point should be expected to be only a little harder than the same steel quenched just below that point, whereas the former will

be fully hardened while the latter will not be sensibly hardened, *if at all.*

The hypothesis, therefore, without any further examination, entirely breaks down. Its working-value is, indeed, very low, and in this respect it is singularly inferior to any of the other theories; for, instead of ignoring the existence of the critical points, these theories have been shaped so as to account for them.

I regret that Mr. Dudley does not present any evidence in favor of his assertion that "Experience does not sustain the conception that steel does not acquire a crystalline structure on a rising temperature." I hope to be able in the near future to take up this question again, and to present fresh experimental evidence in support of my proposition.

Prof. Campbell attacks the question with entirely new methods, and metallurgists will look forward with great interest to the publication of his complete results. It seems best to defer until then a detailed discussion of his views. The question might well be asked at this stage, however, if the fact that the products of the solution of cementite in hydrochloric acid are of a complex nature *necessarily means that cementite itself has a complex chemical structure.* It is mainly upon this inference that Prof. Campbell bases his belief in the existence of a series of ferro-carbons analogous in their constitution to the hydro-carbon series. Let us hope that the chemists will give us their views upon this question. Meanwhile, it must be borne in mind that cementite has been repeatedly isolated by various methods and analyzed by skillful chemists, and that it has always been found to contain iron and carbon in the proportion required by the formula Fe_3C. The inference, therefore, that such is the chemical constitution of cementite is most natural and logical. If metallurgists are told that a view so simple, reasonable and well supported is erroneous, and must now be abandoned in favor of a much more complex structure, they will rightly ask to be shown conclusively that the latter is not a matter of mere speculation; they will ask to be presented with sound and cogent evidences and arguments, which will clearly show Prof. Campbell's hypothesis to be the most probable of the two.

It has now been established quite conclusively that iron free from carbon exhibits the two upper retardations; and, as Mr.

Howe remarks, it is a strong argument in favor of an allotropic change. Indeed, I do not see on what other ground the phenomenon could be accounted for, since it cannot be due to any chemical combination or dissociation. Prof. Arnold's contention that the point A_1 marks the passage of the iron from a plastic to a crystalline condition, if he means, as Mr. Osmond explains, a sudden change in "molecular relation," is, of course, an admission that it is due to an allotropic transformation.

Mr. Howe, who had ascertained, not long since, that a considerable loss of hardening power occurs at the upper retardations, where there is no corresponding change in carbon-condition (a fact difficult for the carbonists to explain), now shows us that iron almost chemically pure (containing only .022 per cent. C) can be enormously strengthened by very sudden cooling, and that this result cannot be due to stress. That such enormous alterations of the physical properties of a nearly chemically pure metal, accompanied as it is by a spontaneous change of its internal energy, must be due to an allotropic change, seems, indeed, an irresistible conclusion.

I believe that upon weighing carefully the evidences presented in this discussion and in previous publications, the impartial student will come to the conclusion that the indications are strongly in favor of some allotropic transformations of the iron taking place at certain critical temperatures.

The question is thus narrowed down to this: (1) Do the allotropic changes contribute to the hardening power of the metal? and (2) if they do, are they the exclusive cause of hardening?

Mr. Howe, in his usual logical and forcible way, argues that both allotropic change and carbon-condition contribute to the hardening of the quenched metal. Allotropic iron, he says, is very strong but not very hard; hence the strength and relative softness of carbonless iron very suddenly cooled, and of manganese-steel, while the hardening carbon confers hardness in the mineralogical sense; hence the very great hardness of quenched high-carbon steel. This, of course, disposes of the objection which has been used repeatedly and with telling effect against the allotropic theory: Why is not manganese-steel intensely hard?

It is interesting to note that Mr. Osmond gets around the difficulty in a very similar way, by assuming that iron in the *gamma*-state, in which condition it is present in manganese-steel, is strong but not very hard, and that mineralogical hardness is due to *beta*-iron.

I somewhat regret that Mr. Osmond has dignified the concluding remarks of my paper with the name of a theory, thus giving them an importance which they did not have in my own mind. His masterly contribution calls for a careful study, and constitutes, I believe, the strongest plea in support of the allotropic theory which has yet been presented.

I can only, for the present, refer somewhat roughly to those of his arguments which have a more direct bearing upon the views expressed in my paper.

I have formulated the following proposition, well supported by experimental evidences: Each critical point is accompanied by a structural change which begins and ends with it. In the ranges of temperatures where there is no critical point, we find no change in the microstructural composition. The accuracy of the first part of this proposition is not questioned by Mr. Osmond. Indeed, it has been shown to be true beyond reasonable doubt. Mr. Osmond, however, thinks that the second part is true only for steel of medium carburization. In very soft steel, and in steel containing over 0.90 per cent. of carbon, he finds that a structural change does take place without corresponding thermal retardation.* The instance which Mr. Osmond gives in the case of soft steel (carbon, 0.14 per cent.) is not conclusive. It is shown that between 1000° and 1340° there is a change in the structural composition of the metal, but it is not shown that such change is not accompanied by a retardation. Indeed, a critical point has been found by Dr. Ball, and I believe by others, in the vicinity of 1300° C. It would seem, therefore, that the evidence in this case strengthens, if anything, my proposition.

In the case, however, of high-carbon steel, heated to a high temperature, Mr. Osmond finds a change of the structural composition taking place very gradually, extending over a considerable range of temperature, and therefore without any critical point. At 800° C., or thereabout, a new constituent appears,

* *Jour. Iron and Steel Inst.*, 1890, No. I.

which increases gradually in quantity, while the free cementite decreases proportionally until it entirely disappears. Certainly it is opposed to that part of my proposition asserting that a structural change is always accompanied by a thermal retardation, and yet I do not believe that its correctness should be doubted too hastily upon the strength of this single instance; at least, not until the appearance and nature of this new constituent has been further investigated. In view of the fact that all the other structural transformations are strictly confined to the ranges of temperature covered by the critical points, I am somewhat reluctant to concede that the appearance of a new constituent, unaccompanied by a retardation, has been conclusively established. I ask for more inquiry.

The mode of occurrence and the properties of austenite, moreover, raise a serious difficulty. A piece of steel containing 1 per cent. of carbon, quenched at a high temperature (say 1000° C.), is made up entirely of martensite (see Mr. Osmond's Fig. 1), and consequently very hard. On the other hand, a piece of steel containing 1.50 per cent. of carbon and similarly treated will contain, roughly, 50 per cent. of martensite and 50 per cent. of austenite. Austenite, Mr. Osmond says, is much softer than martensite, for it can be scratched with a needle. The steel containing 1.50 per cent. of carbon, therefore, should be much softer than the sample containing 1 per cent., while, of course, it is the other way.

Mr. Osmond compares the formation of martensite to a true internal concentration, and it would seem, indeed, as if it were so. And still we must wonder, if such be the case, why the carburization of the ferrite does not proceed gradually and progressively instead of jerkily, occurring only, as it does, during the short ranges of temperature covered by the critical points.

At all events, it remains true that the structural changes and the thermal retardation which occur during the heat-treatment of steel are most closely related. If the former are not the primary cause of the latter (and I do not, by any means, insist that they are), there is a strong presumption that they are a proximate cause. Of course, this does not answer the query, whether the primary cause of the retardations is due to allotropic changes, to chemical changes, or to both—in other words, whether the

solution we seek is to be found in the allotropic, the carbon, or the carbo-allotropic theory. I very frankly admit that the facts that carbonless iron exhibits the upper retardations, and that it can be greatly hardened by sudden cooling, seem to render impossible the hypothesis that the hardening of steel is due to the structural changes studied in this paper. They oppose that hypothesis with as much force as they do the carbon or the carbide theories.

The meaning at present attributed by Mr. Osmond to the critical points differs materially from what had been asserted so far by the allotropists. If I understand him right, it is now urged that in mild steel, where the three critical points occur separately, their significance is as follows:

Above Ar_3, the whole of the iron is in the *gamma* state, associated with the carbon, forming martensite.

During Ar_3, a certain amount of iron is set free, and passes to the *beta* state. The iron of the remaining martensite, however, remains in the *gamma* state. The metal is then made up of structurally free *beta*-iron and of *gamma* martensite.

During Ar_2, an additional amount of iron, previously included in the martensite, is set structurally free, and the whole of the free iron passes to the *alpha* state, the iron of the martensite still remaining *gamma*-iron. The metal is then made up of free *alpha*-iron and of *gamma* martensite.

During Ar_1, the structurally free *alpha*-iron remains unchanged, but the iron of the remaining martensite passes from the *gamma* state to the *alpha* state, simultaneously with the change of the condition of the carbon (or of its distribution?), forming pearlyte.

This new interpretation of the meaning of the critical points constitutes a momentous modification of the allotropic theory, and should be carefully examined.

While, of course, it explains why the martensite found between Ar_1 and Ar_2 is very hard, which could not be the case if its iron were, like the free iron, in the *alpha* state, still one should expect a piece of mild steel (containing, say, 0.22 per cent. of carbon), quenched between Ar_1 and Ar_3, to be much softer than the same steel quenched above Ar_2, seeing that in the former case it contains some 70 per cent. of soft *alpha*-iron, while in the latter it is made up only of martensite. As a mat-

ter of fact it will be somewhat softer, but not by any means as much softer as should be expected.

Again, it is said that there is only *gamma* martensite, and still the magnetic properties of hardened steels of medium hardness, which are made up wholly of martensite, is attributed to the presence of considerable *alpha*-iron. Also, if martensite contains only *gamma*-iron, it should have the same hardness, whatever its degree of carburization. Moreover, it should not be very hard, since *gamma*-iron is not very hard. However, this argument, which would lead us to absurd conclusions if followed to a logical extreme, can probably be refuted satisfactorily by the allotropists on the ground that while martensite *at a high temperature* contains only *gamma*-iron, upon sudden cooling some of its iron will pass to the *beta* and to the *alpha* state, in proportions varying with its carbon-content and the rate of cooling.

It would also seem that a piece of steel quenched between Ar_3 and Ar_2 should contain more *beta*-iron than when quenched above Ar_3, and therefore should have greater mineralogical hardness.

I am far from considering these difficulties fatal to the hypothesis; I merely mention them, so that they can be removed and the way made clear for the allotropic theory, if it be the true one.

The Cement-Materials of Southwest Arkansas.

Discussion of the Paper of Prof. Branner. (See p. 42.)

(Chicago Meeting, February, 1897.)

ROBERT T. HILL, Washington, D. C.: Having studied very minutely the geology of the district referred to by Prof. Branner, I beg to state that his quotation of my classification of the Cretaceous deposits of Arkansas is not the one which I originally gave,* nor is it the one which has since been published and is now generally accepted.† I did not assign any of the Cretaceous formations of Arkansas to the "middle Cretaceous,"

* *Ann. Rep. Geol. Survey of Ark.*, for 1888, vol. ii., pp. 70–152.

† *Bulletin of the Geological Society of America*, vol. 5, pp. 298–338.

but discussed all the formations under two grand divisions as upper and lower.

Neither do I accept the Cretaceous-Tertiary parting, given upon the map (page 45) by Prof. Branner. Not a single undoubted Cretaceous outcrop has been found in Arkansas east of the Iron Mountain railway; and nine-tenths of the line which Prof. Branner has drawn as the Cretaceous-Tertiary parting is covered by extensive sheets of Post-Cretaceous (in part Post-Tertiary) formations, which conceal both the underlying Cretaceous and the Eocene Tertiary formations.

The nomenclature and legend of the geological map accompanying my report was changed without my knowledge by Prof. Branner, who inserted age-names for formation-names in the legend of the map submitted by me, classifying all the various formations under the general names of Cretaceous, Tertiary, etc., in some instances where I had expressly stated in the report that the age of these formations was uncertain.

The subsequent discovery by Prof. Harris of a Cretaceous fossil in the depths of an artesian well at Fulton is not alone sufficient evidence to justify the drawing of a parting line upon the map as Prof. Branner has placed it. Prof. Harris himself has said on page 20 of the report referred to by Prof. Branner, "We have no means of determining the exact location of the Tertiary-Cretaceous boundary line in this part of the State;" and distinctly declares that the boundaries on the map which he gives are his impressions and not a definite parting. The fact of the matter is that nine-tenths of the area which Prof. Branner's map includes under the general name of Cretaceous is occupied by aggradational formations of late Tertiary (Lafayette) and Pleistocene age; and hence if his method of geological mapping by age rather than formation should prevail, the interior Tertiary parting should extend to and in places overlap, the line which he has drawn between the Paleozoic and the Cretaceous.

PROF. BRANNER: It is true that in his Arkansas report Mr. Hill states (p. 188) that the "Lower Cross Timber sands do not crop out in Arkansas." But I have a letter from Mr. Hill dated April 26, 1890 (after his report was out) in which he says: "I found at Pine Bluff on Red river . . . . a grand ex-

posure of the Lower Cross Timber (Dakota) sandstone which proves to be the same character as the Morris Ferry greensands on Little River," Arkansas. I wished to give him credit for this discovery, but I seem to have overdone the matter.

As for the Cretaceous-Tertiary border: Prof. G. D. Harris of Cornell University, one of the best paleontologists in America, studied the Tertiary of Arkansas after Mr. Hill had reported on the Cretaceous. He found the Arkadelphia shales which Mr. Hill called Tertiary, to be Cretaceous, and gave satisfactory evidence of the correctness of that conclusion.* Prof. Harris says the location of the Tertiary-Cretaceous border given on his map is only approximate, and it is so stated in the footnote on the second page of my article.

Mr. Hill claims that I changed formation-names to age-names in the legend of his map. His original map lies before me; the word "formation" is used just once on the original, and that is after the "Trinity."

Inasmuch as these questions have but little to do with the subject of my paper, it is hardly necessary to discuss them further.

The Calorific Value of Certain Coals.

Discussion of the Paper of N. W. Lord and F. Haas. (See p. 259.)

(Lake Superior Meeting, July, 1897.)

WILLIAM KENT, New York City: I regard the paper of Professors Lord and Haas as the most important and valuable that has yet appeared on the subject of the calorific value of American coals. The close correspondence of the results they have obtained by the use of the bomb calorimeter with those calculated by the Dulong formula from the ultimate analyses, confirming the work done by Mahler in France some years before, is evidence of the extreme accuracy of their work, and gives us renewed confidence in Mahler's modification of the Berthelot calorimeter, and in Dulong's formula, which latter, until

* *The Tertiary Geology of Southern Ark.* By G. D. Harris, pp. 15–19.

the publication of Mahler's work, had been somewhat discredited by the researches of Scheerer-Kestner and Meunier-Dollfus in 1868.

The conclusion of the authors that the actual coal (moisture and ash excluded) of a given seam over considerable areas may be regarded as of uniform heating-value, is one of great practical importance. I have held a similar opinion, tentatively, for a long time, as a result of numerous tests of steam-boilers made in different parts of the country during the past fifteen years, and I am glad to have this opinion confirmed by the extensive research of the authors. Should this conclusion be established, or its limitations defined, by future tests, it will be possible for us to approximate closely the heating-value of any given sample of coal by ascertaining where it is mined and by determining its moisture, sulphur and ash, which are the three variable elements in lots of coal from the same mine, without going to the expense of an ultimate analysis or a calorimeter test. In any given mine or seam the sulphur, averaged from car-load lots, is reasonably constant, especially in such coals as are of good repute in the market as steam-coals. The moisture and ash, however, are subject to accidental variations, but they are easily determined.

While thus commending the paper as a whole, I have a few minor criticisms to make upon it. I do not think it is right to include in the Dulong formula the term + 2250 S for the sulphur. The sulphur does not exist as free sulphur, but as pyrites, with possibly some sulphates, and it can have little or no heating-value. Taking the average calorific values as found by the calorimeter and by calculation from analysis, I find that the latter is in excess by 21 calories. If the authors had used Berthelot's recent figure, 8140, for the heating-value of carbon, instead of the old figure of Favre and Silberman, 8080, their calculated average result would be about 36 calories higher, or 57 calories higher than the result obtained by the calorimeter. The sulphur, averaging 1.68 per cent., gives, if we allow 2250 as its heating-value, about 38 calories. Subtracting this from 57 gives 19 calories excess. That is, if the formula used had been Cal. power $= 8140\ C + 34{,}462\ (H - \frac{1}{8} O)$, the calculated average result would have been only 19 calories higher than the average result obtained by the calorimeter tests. This

figure is surprisingly close to the average result obtained by Mahler, as shown in Table I., below. Their calculated average result for four classes of coals (31 tests in all), ranging from 97 down to 59 per cent. of fixed carbon in the combustible, is 18 calories lower than the result obtained by the calorimeter. Taking the results of Mahler together with those of Professors Lord and Haas, the average of all shows a practically absolute agreement of the calculated with the calorimeter figures.

The form in which the authors have reported their results is not as convenient for comparison as the form used by Mahler. Columns should have been given showing the percentage of fixed carbon in the "combustible," or "coal dry and free from ash," and the heating-value per pound of this combustible. I have calculated these values, and present them in Table II. below. By inspection of this table it is seen how remarkably uniform is the constitution and the heating-value of the combustible portion of the different samples of coal from the same seam, although mined over a considerable area of country.

I have discovered, in the analyses given in the paper, an interesting relation between the percentage of carbon as found by the ultimate analysis and the percentage of fixed carbon as found by the proximate analysis. It is, that in the bituminous coals the fixed carbon is nearly equal to the total carbon minus five times the available hydrogen ($H - \frac{1}{8} O$), and that in the semi-bituminous coal (Pocahontas) it is nearly equal to the total carbon minus three times the available hydrogen. The following is the calculation from the average analysis:

	Available H.	Total C.	Difference.	Fixed carbon.	Difference.
Upper Freeport,	$3.94 \times 5 = 19.70$	72.65	52.95	51.63	— 1.32
Pittsburgh,	$4.15 \times 5 = 20.75$	75.24	54.49	53.81	— .68
Darlington,	$4.01 \times 5 = 20.05$	75.19	55.14	54.69	— .45
Hocking Valley,	$3.34 \times 5 = 16.70$	68.03	51.33	49.64	— 1.69
Thacker,	$4.27 \times 5 = 21.35$	78.65	57.30	56.67	— .63
Mahoning,	$3.71 \times 5 = 18.55$	71.13	52.58	50.95	— 1.63
Pocahontas,	$3.89 \times 3 = 11.67$	84.87	73.20	74.84	+ 1.64

These figures indicate that in the bituminous coals the volatile hydrocarbon (excluding H_2O) is equivalent to $2C_2H_4 + CH_4$, or to 5 parts C and 1 part H; and that in the semi-bituminous coals the volatile hydrocarbon is equivalent to CH_4, or

TABLE I.—*Heating-Power of Coals.* [P. Mahler.*]

		COAL DRY AND FREE FROM ASH.						
	ANTHRACITE AND ANTHRACITIC.	Percentage of Fixed Carbon.	COMPOSITION.			HEATING-POWER CALORIES.		
			C.	H.	O+N.	Actual.	By Dulong's† Formula.	Difference.
1	Pennsylvania	97.00	95.37	2.20	2.43	8256	8462	+ 206
2	De la Mure (Grand Couche)	97.25	95.24	1.50	3.26	8216	8178	— 43
3	Hay-Duong (Tonkin)	96.83	92.86	2.16	4.99	8121	8130	+ 9
4	Kebao	94.80	93.46	3.07	3.48	8532	8528	— 4
5	Commentry	96.81	91.49	3.12	5.39	8456	8333	— 123
6	Blanzy, Puits Ste-Barbe	94.00	90.00	3.17	6.83	8203	8169	— 34
7	Grande Combe, Puits Petassas	93.29	91.46	3.95	4.59	8540	8653	+ 113
8	Creusot	89.56	92.89	3.78	3.33	8687	8704	+ 17
	Average							+ 18
	FAT AND SEMI-FAT (DEMI-GRASSE).							
9	Demi-Grasse, d'Anzin, Fosse St. Marc	85.92	91.26	4.27	4.48	8656	8751	+ 95
10	" Grande Combe	86.62	91.19	4.46	4.35	8756	8817	+ 61
11	" Roche-la-Molière	86.00	90.11	4.38	5.51	8767	8651	— 116
12	" Aniche	88.07	90.10	4.40	5.49	8834	8659	— 175
13	Grasse, Anzin, great vein	78.49	89.20	4.67	6.14	8574	8651	+ 77
14	" Ronchamp	76.77	88.89	4.84	6.27	8797	8678	— 119
15	" Lens	80.50	90.08	4.60	5.17	8839	8805	— 34
16	" Carmaux	78.25	87.84	4.87	7.30	8639	8559	— 80
17	" Roche-la-Molière	77.15	89.53	4.84	5.63	8867	8757	— 110
18	" Saint Etienne	79.16	89.23	5.03	5.74	8857	8796	— 61
19	" Mines de Portes (Gard.)	80.71	86.52	4.84	8.64	8667	8382	— 285
	Average							— 66
	FAT GAS COALS.							
20	Bethune	69.59	87.08	5.37	7.60	8668	8654	— 14
21	Lens	69.20	87.26	5.44	7.30	8748	8705	— 44
22	Firminy	67.98	85.39	5.58	9.13	8578	8524	— 49
23	Montrambert	65.73	84.52	5.54	9.94	8598	8407	— 191
24	Commentry	60.04	85.86	5.60	8.73	8408	8573	+ 165
25	Wigan, Lancashire	68.36	88.57	5.72	5.72	8768	8979	+ 211
26	Cannel coal, Niddrie	47.00	88.79	6.57	9.63	8431	8717	+ 286
	Average							+ 52
	FLAMING COALS, LIGNITIC.							
27	Montoie	62.93	88.95	5.64	10.42	8570	8371	— 199
28	Blanzy (Puits Ste.-Marie)	68.05	84.26	5.27	10.46	8350	8271	— 79
29	Decazeville (Bourran)	64.20	83.17	5.68	11.14	8270	8294	+ 24
30	Blanzy (Puits Ste.-Eugénie)	60.61	81.54	5.64	12.83	8083	8072	— 11
31	Decazeville (Tramont)	58.77	78.72	5.67	15.61	7837	7735	— 102
	Average							— 74
	Average of above four classes							— 18
	LIGNITES.							
32	Terre de Feu	47.23	71.01	5.94	23.05	7039	6882	— 157
33	Trifail (Styria)	49.66	69.24	5.06	25 71	6616	6317	— 299
34	Vaurigard	50.05	66.36	5.01	28.63	6076	5938	— 138
35	TURF FROM BOHEMIA	31.07	57.21	5.96	36.82	5903	5169	— 734
	WOOD.							
36	Partially dry, Sapin de Norvège		51.08	6.02	42.90	4828	4428	— 400
37	Bois de Chêne de Lorraine		50.44	5.88	43.69	4689	4293	— 396
38	Cellulose, $C_{12}H_{10}O_{10}$		44.44	6.17	49.39	4200	3617	— 583

* *Bulletin de la Société d'Encouragement pour L'Industrie Nationale*, 1892. Also, Rothwell's *Mineral Industry*, 1892, vol. i., p. 97.

† Dulong's formula, as given by Mahler, is:

$$Q = \frac{1}{100}\left[8{,}140C + 34{,}500\left(H - \frac{(O+N)-1}{8}\right)\right]$$

TABLE II.—*Heating-Power of Coal.*

[Calculated from Lord and Haas's Results.]

No.	POCAHONTAS, VA	COAL DRY AND FREE FROM ASH.	
		Per Cent. Fixed Carbon.	Calorific Value.
1	Run of mine	80.10	8768
2	" "	81.19	8731
3	" "	80.14	8732
4	" "	80.61	8745
5	" "	80.29	8777
	Average	80.48	8751
	MIDDLE KITTANNING, HOCKING VALLEY, OHIO.		
6	Lump	57.54	7870
7	"	56.14	7913
8	Run of mine	59.20	7762
9	" "	57.68	7797
10	Lump	57.64	7767
	Average	58.12	7822
	THACKER COAL, WEST VIRGINIA.		
11	Run of mine	62.00	8434
12	" "	61.77	8513
13	Nut	60.59	8425
14	"	60.75	8496
	Average	61.28	8467
	PITTSBURGH COAL, PENNSYLVANIA.		
15	Carnegie, Allegheny Co	60.68	8304
16	Turtle Creek, "	62.14	8378
27	Carnegie, "	59.30	8352
28	" "	57.99	8248
29	Creedmore, "	56.79	8324
34	N. Mansfield, "	59.26	8277
38	Turtle Creek, "	59.42	8289
	Average	59.39	8313
	MIDDLE KITTANNING (DARLINGTON COAL), PENNA.		
17	Hoytdale, Lawrence Co	61.30	8278
18	Beaver Creek, "	61.75	8201
19	Wampum, "	59.14	8256
20	Near Wampum, "	60.28	8244
32	Hoytdale, "	60.38	8177
33	Wampum, "	57.56	8267
35	Clinton, "	60.18	8166
	Average	60.09	8226
	UPPER FREEPORT COAL, OHIO AND PENNSYLVANIA.		
21	East Palestine, Ohio	60.08	8113
22	" "	57.81	8257
23	Waterford, "	58.85	8230
24	Yellow Creek, "	56.79	8330
25	Steubenville, "	56.78	8267
26	Cambridge, "	57.13	8041
30	Steubenville, "	55.71	8037
31	Salineville, "	59.26	8160
36	Palestine, "	58.01	8339
37	New Galilee, "	58.76	8165
40	Palestine, "	59.01	8224
	Average	58.02	8197
	MAHONING COAL.		
39	Salineville, Ohio	59.28	8182

3 parts C and 1 part H. If these relations should be confirmed by other coal analyses, they may be useful as a criterion of the accuracy of the proximate analysis. Also having the ultimate analysis of a coal, and knowing its class, as bituminous or semi-bituminous, the proximate analysis may be calculated therefrom with slight probability of error.

I quote the following extracts from the paper (pp. 268, 269):

"A relation between the fixed carbon and the calorimetric test was stated by Mr. Kent ('Heating-Value of Coal,' *Min. Ind.*, 1892, p. 97); but the results of our work do not appear to correspond to his figures. Taking the Pittsburgh coal, we find the average calorific power to be 7532. . . . . Interpolating from Kent's table would give 8054, a difference of 259 units, or 3.2 per cent. The average Freeport coal shows a difference of 296 units, or nearly 4 per cent. The determination of fixed carbon is very uncertain, being much influenced by slight changes in method; therefore it is entirely possible that these differences are due to our method of analysis, giving low results as compared with that used by the chemists furnishing his figures. . . . . Our determinations of fixed carbon could not be used for estimating the calorific power within any satisfactory limit of accuracy."

"Attempts to derive a general law for all the coals examined were abandoned."

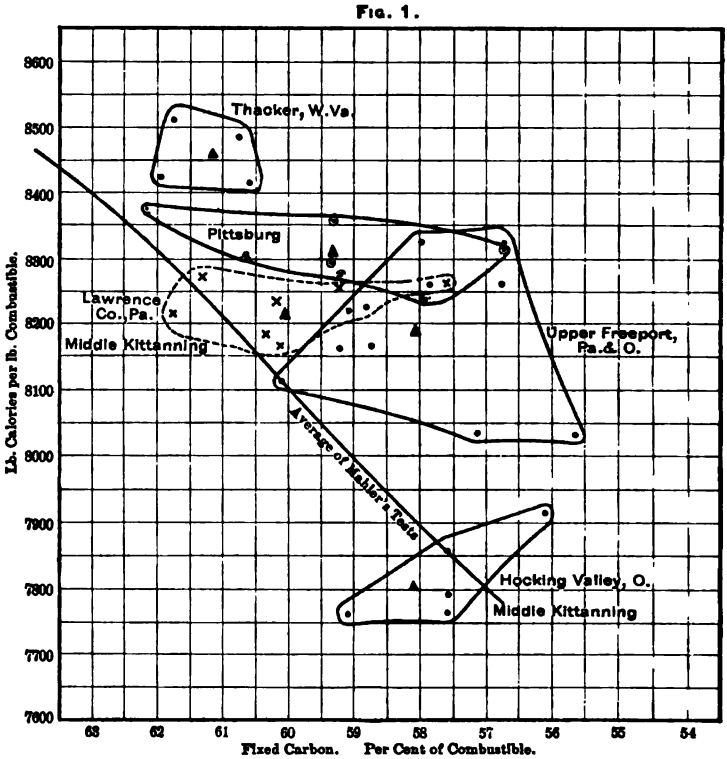
These statements of the authors refer to the curve which I plotted from Mahler's tests of European coals, published in my article in vol. i. of *Mineral Industry*, and to the table which I derived therefrom. The table is as follows:

Approximate Heating-Value of Coals.

Per Cent. of Fixed Carbon in Coal Dry and Free from Ash.	Heating-Value.		Per Cent. of Fixed Carbon in Coal Dry and Free from Ash.	Heating-Value.	
	Calories.	British Thermal Units.		Calories.	British Thermal Units.
97	8200	14,760	63	8400	15,120
94	8400	15,120	60	8100	14,580
90	8600	15,480	57	7800	14,040
87	8700	15,660	54	7400	13,320
80	8800	15,840	51	7000	12,600
72	8700	15,660	50	6800	12,240
68	8600	15,480			

The value of my curve and table for use in connection with American coals having been thus called in question, I have been led to study the subject anew, with the view of comparing the work of the authors with that of Mahler and of learning whether or not there was any essential difference between the American and European coals, and whether a curve plotted from tests of the latter would be of any value when applied to the former.

Fig. 1.

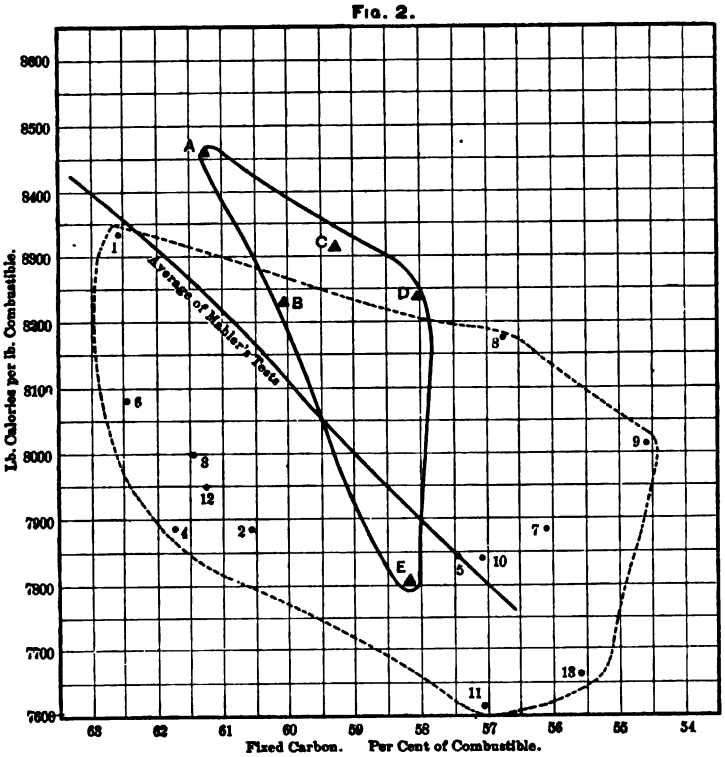


Relation of Heating-Value to Per Cent. of Fixed Carbon.
(Tests of Profs. Lord and Haas.)

In Fig. 1 I have plotted a portion of the curve derived from the results of Mahler between the limits of 55 and 63 per cent. of fixed carbon, together with the results obtained by Lord and Haas on coals lying within the same limits. The dots and crosses represent the individual tests, and the small black triangles the average figures for each class of coal. Each class of coal is surrounded by a boundary line showing the extent of variation, or what I call the "field" of each coal.

In Fig. 2 the same portion of the Mahler curve is given, with the average results of Lord and Haas, together with the results of calorimeter tests of thirteen different varieties of coal,

FIG. 2.



Relation of Heating Value to Per Cent. of Fixed Carbon.

Numbered points are tests by C. W. Houghton. Lettered points are averages of Lord and Haas's tests. The several coals are as follows:

A, Thacker, W. Va.;
B, Lawrence Co., Pa.;
C, Pittsburgh, Pa.;
D, Upper Freeport, Pa.;
E, Hocking Valley, O.;
1, Youghiogheny, Pa.;
2, Pittsburgh, Pa.;
3, Vanderpool, Ky.;
4, Brier Hill, O.;
5, Hocking Valley, O.;
6, Big Muddy, Ill.;
7, Streator, Ill.;
8, Ladd, Ill.;
9, Seatonville, Ill.;
10, Wilmington, Ill.;
11, Mt. Olive, Ill.;
12, Indiana Block;
13, Indiana Lump.

which were made for me last year by Mr. C. W. Houghton, M.E., assistant in Sibley College, Cornell University, using the coal-calorimeter of Prof. R. C. Carpenter, which is described in the *Transactions* of the American Society of Mechanical En-

gineers, vol. xvi., p. 1040. The results of these tests were as follows:

	Coal Dry and Free from Ash. Fixed Carbon. Per cent.	Heating Value. Calories.
1 Youghiogheny, Pa.,	62.6	8330
2 Pittsburgh, Pa.,	60.6	7890
3 Vanderpool, Ky.,	61.5	8000
4 Brier Hill, O.,	61.8	7890
5 Hocking Valley, O.,	57.5	7830
6 Big Muddy, Ill.,	62.5	8080
7 Streator, Ill.,	56.2	7890
8 Ladd, Ill.,	56.8	8170
9 Seatonville, Ill.,	54.7	8060
10 Wilmington, Ill.,	57.1	7840
11 Mt. Olive, Ill.,	57.1	7610
12 Indiana Block,	61.4	7950
13 Indiana Lump,	55.6	7670

The figure for Streator coal is the average of five tests of samples from as many different car loads, the range being from 7780 to 8000 calories. The figure for Big Muddy is the average of two lots, which varied 170 calories; and that from Wilmington is the average of two lots which varied 80 calories. The other tests were of only one sample each.

The plotting of these tests shows that they cover quite a wide field, and tends to confirm the conclusion of the authors that the heating-power has no definite relation to the fixed carbon; but it will be observed that the general trend of the field of Houghton's tests is in the direction of the Mahler line; that the maximum deviation of any single test (except No. 9) from the Mahler line is less than 500 calories, or about 6 per cent.; and that Houghton's tests arrange themselves about equally on each side of the Mahler line. I cannot vouch for the accuracy of Mr. Houghton's figures either as to percentage of fixed carbon or as to heating-value. The tests were simply commercial ones, made to check the results of boiler tests. I am inclined to believe that the figures of heating-value are much more reliable than those of the percentages of fixed carbon, for the latter, as is said by Profs. Lord and Haas, is not easy to determine with accuracy.

Fig. 1 is especially interesting in showing that each of the five classes of coals tested by the authors of the paper, within the limits of 55 and 62 per cent. of fixed carbon, has a law of

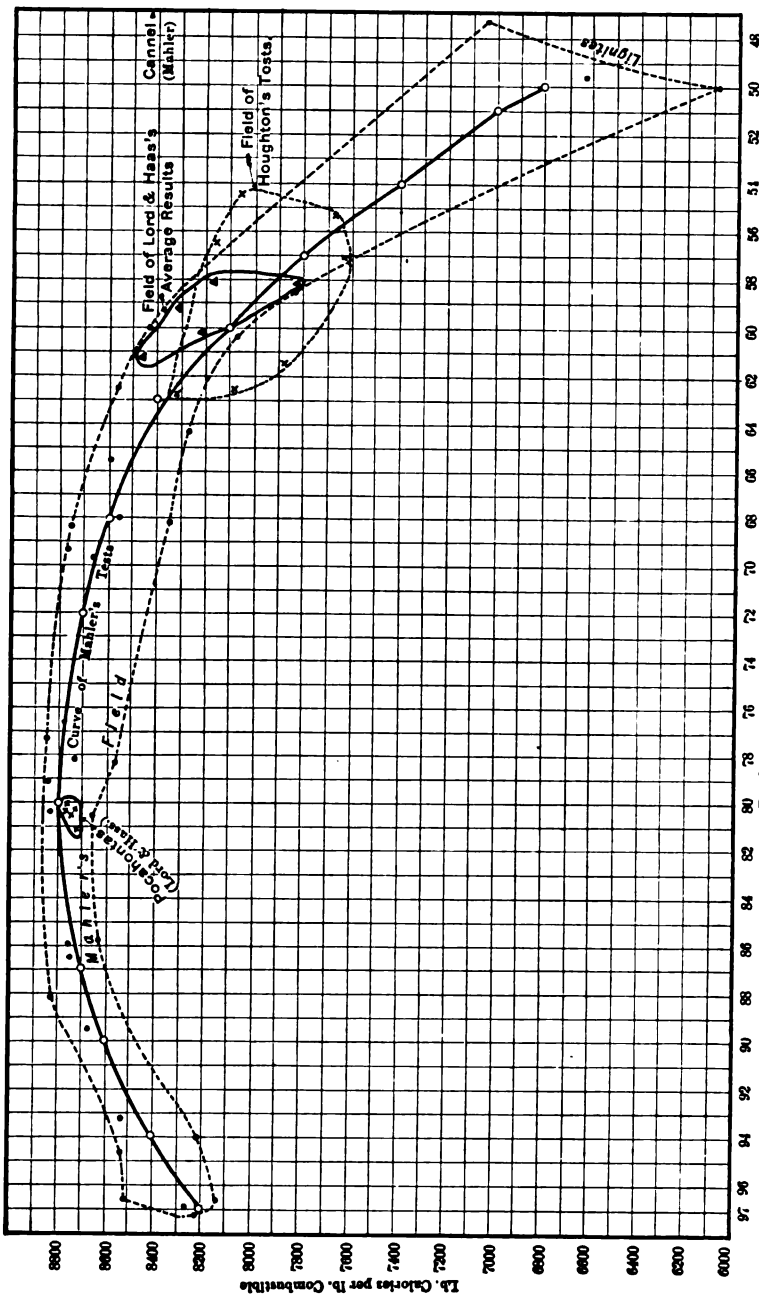


FIG. 3.

Relation of Heating-Value of Coal to Per Cent. of Fixed Carbon in the Combustible.
(Results of Mahler, Lord and Haas and Houghton, compared.)

its own. Three of these classes, the Pittsburgh, the Lawrence Co., Pa., and the Hocking Valley, O., seem to have a very uniform heating-power through a wide range of variation in percentage of fixed carbon. The same might be said of the Upper Freeport coal, if the two tests which give less than 8100 calories were omitted. The four samples of Thacker, W. Va., coal are very close together, both in heating-power and in percentage of fixed carbon.

In Fig. 3 are plotted all of Mahler's results between the limits of 47 and 97 per cent. of fixed carbon. A boundary line is run around all the tests, and the average curve which I gave in the paper in *Mineral Industry* is reproduced. Comparing the curve with the plottings of the individual tests, and with the boundary enclosing them, it will be seen that they justify the conclusion stated in that paper, viz., that "knowing the percentage of fixed carbon in the dry coal free from ash, we may, in the case of all coals containing over 58 per cent. of fixed carbon, predict their heating-value within a limit of error of about 3 per cent."

Fig. 3 shows also that the figures for Pocahontas coal obtained by Lord and Haas all come remarkably close to the Mahler line, all five tests lying entirely within the Mahler field. The average figure from these tests, 8751 calories, is only 49 calories, or less than 0.6 per cent., lower than the figures in my table for 80 per cent. of fixed carbon.

Another thing shown by Fig. 3 is, that Lord and Haas's tests cover only a small portion of the range of composition of the coals tested by Mahler. Mahler's tests, excluding the lignites, cover the entire range between 58 and 97 per cent. of fixed carbon, while Lord and Haas's are confined between 55.7 and 62.2 per cent., except the five tests of Pocahontas coal, which are between 80.1 and 81.2 per cent.

With the three diagrams, Figs. 1, 2, and 3, we may find what is the probable error of the conclusion that I drew five years ago from the study of Mahler's work, viz., that "knowing the percentage of fixed carbon in the dry coal free from ash, we may, in the case of all coals containing over 58 per cent. of fixed carbon, predict their heating-value within a limit of error of about 3 per cent." Excluding the coals that have below 58 per cent. of fixed carbon in the "combustible," the variation of any one of

Lord and Haas's coals from the Mahler line does not exceed 320 calories, or 4 per cent. Taking the average figure for each class of coals, it falls in all cases within the limit of 3 per cent. The figures from Houghton's tests also fall within the limit of 4 per cent. variation from the Mahler line, except coal No. 4, Brier Hill, O. (of which only one test was made), which falls 400 calories, or nearly 5 per cent., below the Mahler line.

On the whole, therefore, I consider that both Lord and Haas's and Houghton's tests are a substantial confirmation of the conclusion I drew from Mahler's tests. Taking into consideration the fact that the reported percentage of fixed carbon is very apt to be 2 or 3 per cent. in error, I am disposed to hold to my original conclusion, at least until a larger series of tests may show that it should be modified.

It is to be observed, however, that the Mahler line falls rapidly with percentages below 62 per cent. of fixed carbon; and it is therefore to be expected that below this point there will be a greater range of variation in heating-value than above it. When the volatile matter exceeds 38 per cent., an increasing proportion of it is oxygen, and the relative proportion of oxygen in the highly volatile coal varies in the coals of different districts, as is shown by Lord and Haas's analyses. Thus the Upper Freeport coal averages only 9.58 per cent. of O (in the coal dry and free from ash), while the Hocking Valley coal averages 16.10 per cent., although both coals have the same percentage of fixed carbon, viz., 58 per cent. Full credence, therefore, is to be given to the conclusions drawn from Lord and Haas's tests, that, when the fixed carbon is less than 62 per cent. of the combustible, each class of coal has a law of its own, and coals of any one class may differ in heating-power from the coals of another class containing the same percentage of fixed carbon to an extent as great as 5 per cent.—as in the case with the Upper Freeport and the Hocking Valley coals.

If I rightly understand the ultimate analyses given by Professors Lord and Haas, the hydrogen and oxygen reported include the moisture, as well as the hydrogen and oxygen combined with carbon in the coal. I judge this from the fact that the percentage of ash in the ultimate analysis is the same as that in the proximate analysis of the moist coal. The calcu-

lated calorific power also appears to be that of the moist coal. This is an unusual method of reporting the ultimate analysis of coal. Two other methods are usually followed: 1, that of reporting the complete ultimate analysis of the moist coal, but distinguishing between the moisture which may be driven off by moderate heating and the H and O which are combined with carbon; 2, that of reporting the moisture and the ultimate analysis of the dry coal separately. Taking the average Hocking Valley coal, the three methods of reporting are shown below:

	Method of the paper.	Usual methods. Moist coal.	Dry coal.
Carbon,	68.03	68.03	72.84
Hydrogen,	5.29	4.56	4.88
Oxygen,	15.64	9.78	10.47
Nitrogen,	1.44	1.44	1.54
Sulphur,	1.59	1.59	1.70
Ash,	8.00	8.00	8.56
	99.99	.....	
Moisture,	.....	6.59	.....
		99.99	99.99
Moisture, given separately,			6.59

The last method appears to be the most convenient for studying the characteristics of different kinds of coal. The figures for hydrogen and oxygen in the second column are derived from those in the first by subtracting $\frac{1}{9}$ and $\frac{8}{9}$, respectively, of 6.59 from the H and O. The figures in the third column are derived from those in the second by dividing the latter by (1 — .0659) or .9341.

I trust that Professors Lord and Haas will continue their researches on American coals, extending them to coals of other districts and of a greater range of composition than those in their present paper. In such future research it need not be necessary to go to the trouble and expense of making an ultimate analysis of each coal. A proximate analysis and a determination of the heating-power by means of the bomb calorimeter should give all the needed data. I commend Mahler's words in this connection: "It is difficult to determine with precision the hydrogen and the carbon contained in a coal; it is infinitely more simple to have recourse to a calorimeter, which permits us to appraise the value of all the combustibles without exception, and with incomparable precision."

I wish to call attention to the fact that the determinations of moisture in bituminous coals hitherto published have usually been largely in error. Drying for an hour at a temperature but a trifle above 212° F. will not drive out all the moisture. Coals thus dried and reported to contain 8 per cent. of moisture have actually contained as high as 14 per cent. It is necessary to dry such coals for a long time at from 240° to 280° F., and until repeated weighings begin to show an increase in weight due to oxidation. There need be no fear that such drying will drive off volatile matter. No appreciable amount of it is driven off at temperatures below 350°; at least I found this to be the case with the coals whose heating value was determined by Mr. Houghton, mentioned above.

PROF. N. W. LORD, Columbus, O.: I have been much interested in Mr. Kent's valuable discussion and amplification of the paper by Mr. Haas and myself.

His comparison between the available hydrogen and the fixed carbon as well as his retabulation of our results extend the application of the work. I wish, however, to note a few points in justification or explanation of some of the matters he criticises in the paper.

The statement that the sulphur existing as pyrites can have little or no heating-value, I think is hardly warranted. The experience of "pyritic smelting" as well as the results of pyrites roasters certainly show the fuel-value of sulphides. The coals of Ohio contain their sulphur largely in the form of pyrites, in some cases with quite a noticeable percentage in the form of organic sulphur compounds. The amount present as sulphates is very trifling. I have tested several coals for sulphates, and rarely found more than a trace. I made two experiments last spring, which may be of interest as measuring the fuel-value of pyrites.

A sample of Pocahontas coal (No. 1 in our paper) having a calorific value of 8062 and containing 0.57 per cent. of sulphur, was mixed with pyrites in two proportions, nine of coal to one of pyrites, and eight of coal to two of pyrites. The coal and pyrites were separately reduced to fine powder and then mixed by rubbing in a mortar. The mixtures were then compressed into cylinders for combustion in the bomb. The pyrites used was a selected crystal of FeS_2.

The results were as follows:

No. 1. Weight of coal mixture taken, . . . 0.938 gramme.
Actual heat developed after correction for wire burned as fuse, ... 7129 units.

To calculate the heat due to the production of nitric acid I subtracted the acidity due to the sulphuric acid produced from the total acidity in the bomb-washings and figured the difference as nitric acid. Correcting the figure for heating-power for this gives 7107 units for the heat produced by burning the 0.938 gramme of mixture, but this mixture contained $\frac{9}{10}$ coal. The heating-value of this coal ($0.9 \times 0.938 \times 8062$) was 6806. $7107 - 6806 = 301$, the heat due to the combustion of the pyrites.

The sulphur was determined in the liquid washed from the bomb. It amounted to 0.0538 gramme; or, deducting the 0.0048 in the coal present, 0.0490 sulphur burned as pyrites produced 301 units, which is in the proportion of 6140 units for each unit of sulphur present as iron pyrites in the coal.

A second experiment was conducted in precisely the same manner on the mixture containing eight parts of coal and two parts of pyrites.

	H. U.
Heat of combustion of 0.931 gramme of the mixture, . .	6520
Heat due to coal, ...	6005
Heat due to pyrites, ...	515

Sulphur in bomb-washings as sulphates, 0.1045; present in coal, 0.0042; burned in pyrites, 0.1003; hence this experiment gave 5150 units for the heat due to a unit of sulphur as pyrites.

Of course, these two results do not "check" very well, as all the errors of the test accumulated in the differences found; but it seems safe to conclude that the heat due to the combustion of pyrites in the bomb is somewhere about 5500 units per unit of sulphur. Of course the sulphur is here burned to SO_3, or rather to dilute H_2SO_4, and gives more heat than when it burns in air to SO_2. Pyrites contains 53.3 per cent. sulphur. Translating the above result (5500) into heat developed per unit of FeS_2 gives 2931 heat units.

Berthelot gives for the heat of formation of dilute H_2SO_4 what is equivalent to 4388 units per unit of sulphur; and assuming 1582 as the heating-value of iron burned to magnetic oxide (Andrews), a calculation for the heating-value of pyrites would give:

0.533 S, ...	2339
0.467 Fe, ...	739
Calculated heat, ...	3078

The S being burned to dilute H_2SO_4.

This corresponds to the value found well enough to show that when pyrites burns, the iron and sulphur give nearly the same heat they do when *burned separately in the free state*, which justifies the introduction into Dulong's formula of the sulphur term. As to the number I have adopted in the formula (2250) for the heat developed when S burns to SO_2, it was taken as an average of several published figures, and is probably a little too high, but not enough out of the way to affect the results noticeably, especially as the heat due to the combustion of the iron was omitted, which it would appear should have been included, though it would have amounted to very little.

Mr. Kent is correct in supposing that the analyses as reported included the hydrogen and oxygen of the moisture. This seems to me the correct way. I am always in favor of reporting an analytical result just as it is obtained; it can easily be "figured on" afterward. I think the last point noticed by Mr. Kent is a justification of this course, where he calls attention to the great uncertainty in moisture-determinations. If published moisture-determinations are "largely in error" is it not better to give the ultimate analysis as it is found, and then let the moisture-determination stand on its own merits? Why reduce a coal to a "dry basis" when we cannot be certain what the dry basis is?

In concluding I would say that I am willing to grant the general relation between fixed carbon and heating-value in the coals having high fixed carbon; but for such coals as the Ohio mines furnish, the relation is, as Mr. Kent well says, such that "each class of coal has a law of its own," which would correspond exactly with the general conclusions of the paper.

The Electrolytic Assay as Applied to Refined Copper.

Discussion of the paper of Mr. G. L. Heath. (See p. 390.)

(Lake Superior Meeting, July, 1897.)

ERWIN S. SPERRY, Bridgeport, Conn.: The analysis of refined copper is a subject of great importance, and has not received the attention it deserves. Copper metallurgists, therefore, will welcome the paper of Mr. Heath with satisfaction. The state of the art of the chemical analysis of copper has been such that consumers, if they could not afford to run any risk, have been obliged to buy copper from the smelter having the best reputation, and a specification for copper to meet certain requirements could not be drawn up by the mere giving of the chemical constituents. Mr. Heath has brought the analysis of the copper to such a state that when a specification calls for copper "equal to Lake copper" one will know immediately what percentage of the pure metal to expect.

It seems to be a pretty well established fact that the copper of Lake Superior is practically free from antimony and bismuth; and, with the exception of arsenic, the other elements are easily separated from copper by electro-deposition in an acid solution. The authorities differ in regard to the question whether antimony is deposited with the copper in an acid solution. Classen* recommends the employment of a solution containing 10 per cent. of free nitric acid, sp. gr. 1.21, and the strength of the current from 0.3 to 0.4 ampère, in order to prevent the other metals from being deposited with the copper; and for this reason, I presume, the method has been generally used. I, myself, have been accustomed to use such a solution for the analysis of copper-alloys, but have found that for such alloys only a small quantity of nitric acid is necessary, if sulphuric acid is present in sufficient amount to give the solution the required conductivity.

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* *Quantitative Analysis by Electrolysis*, N. Y., 1888, p. 66.

E. F. Smith* says that arsenic can be previously removed from a copper-solution by evaporation with bromine. By analogy, antimony would probably be likewise volatilized; but no experiments or references to bear out this point are given. If such is the case under all conditions, it is an important fact, as an assay can be made of copper containing a large amount of antimony or arsenic by first removing them by bromine, and then electrolyzing. A searching investigation upon this point is of importance. I should be glad to hear whether Mr. Heath has made any experiments along this line.

The fact that antimony is deposited with the copper was noticed by Hampe.† He found that electrolytic copper contained from 0.007 per cent. to 0.02 per cent. of antimony, and this led him to believe that antimony is carried down with the copper. He accordingly carried out the following experiments:

A sample of copper sulphate containing 25.09 per cent. of copper and 0.0083 of antimony was electrolyzed by a current from six Meidinger-Pincus cells. The deposited copper contained 0.007 per cent. of antimony. Next, an alloy of 99.44 per cent. of copper and 0.529 per cent. of antimony was dissolved in nitric acid, evaporated with sulphuric acid until fumes were given off, the residue of antimonic acid filtered off, and the solution then electrolyzed after adding 10 per cent. of dilute nitric acid (sp. gr., 1.21). The deposited copper contained 0.0198 per cent. of antimony. For this experiment he used a large amount of alloy, 50 grammes.

From these experiments it would appear that, if antimony is present in the copper solution, it is impossible to prevent its simultaneous deposition with the copper. Bismuth is apparently deposited in the same manner. The copper with which Mr. Heath has to deal can be accurately assayed by the electrolytic process; but if antimony or bismuth are present, the determination of these elements as well as that of the percentage of copper is of vital importance; the assay by the electrolytic method, therefore, is not suited for such copper.

The injurious effect of antimony and bismuth on brass is not

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* *Electro-Chemical Analysis*, Phila., 1890, p. 93.

† *Chemiker Zeitung*, xvi., 1892, pp. 417, 418.

generally appreciated, *but the difference between good and poor brass often lies in the presence or absence of* 0.02 *per cent. of antimony;* so that it can readily be seen how important the determination of these elements is. The freedom of Lake copper from antimony and bismuth, and their presence in electrolytic material, easily shows that the reputation of Lake brands as the most suitable material for the manufacture of high-grade brass and German silver has been justly earned.

For the determination of the arsenic, antimony and bismuth I have found the method of Rivot* to be the most satisfactory; in fact, it is the only method known which is really suitable for copper containing very small amounts of these elements. By it one is enabled to employ a large amount of copper for analysis, precipitate it by potassium sulphocyanide, and determine the impurities in the filtrate. The copper in such material is best determined by difference.

I agree with Mr. Heath that copper can be deposited in a sulphuric acid solution, and have repeatedly carried out such a deposition; but the solution should contain enough acid to give it the proper conductivity. The deposited copper is of a dark red color, and has not as clean and pure an appearance as when a little nitric acid is present. Even a few drops of nitric acid is all that is necessary to accomplish this result. As Hampe† has shown that nitric acid does not prevent antimony from coming down, and as bismuth is likewise deposited, the question occurred to me whether nitric acid was necessary in a large amount to prevent the co-precipitation of such metals as zinc. To investigate this point the following experiments were carried out:

A sample of brass made from Lake copper and pure spelter was rolled to No. 30 gauge in order to render it homogeneous, and three determinations of the copper were made. One gramme was used in each case.

A.—Dissolved in nitric acid; added enough sulphuric acid to convert the zinc and copper into sulphates, but to leave no excess. Evaporated until all the nitric acid was removed, and then added 8 c.c. of nitric acid of sp. gr. 1.21. Diluted until the volume was 80 c.c. and electrolyzed with a current of 0.3

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* *Comptes Rendus*, xxxviii., p. 868. † *Loc. cit.*

ampère. Allowed the deposition to take place until all the copper was deposited, as shown by testing with hydrogen sulphide. Washed with water, and then with alcohol. Dried by igniting the alcohol.

B.—Dissolved in nitric acid and evaporated with 5 c.c. of concentrated sulphuric acid, sp. gr. 1.8, until the nitric acid was removed. Diluted to 80 c.c., added 5 drops of nitric acid, and then electrolyzed as before.

C.—Dissolved in nitric acid, evaporated with 2 c.c. of concentrated sulphuric acid until the nitric acid was removed. Diluted to 500 c.c.; added 50 c.c. of hydrochloric acid of sp. gr. 1.1, and precipitated the copper by hydrogen sulphide. Filtered, with the usual precautions, and then dissolved in nitric acid, evaporated with 2 c.c. of concentrated sulphuric acid, added 4 c.c. of nitric acid, sp. gr. 1.21, and electrolyzed. Treated the copper as before.

The results obtained were as follows:

	A.	B.	C.
Copper, per cent., . . .	63.355	63.342	63.312

These results show that a large amount of nitric acid is unnecessary in the analysis of the alloys of copper and zinc, and that a sulphuric acid solution is satisfactory. There is one point to be taken into consideration, however, namely, the disadvantage of considerable sulphuric acid in the solution remaining after the deposition of the copper. If such a solution is to be evaporated to dryness, a nitric acid solution such as was employed in experiment A above is the most suitable. In the assay of copper, as described by Mr. Heath, the sulphuric acid solution of the strength mentioned by him certainly offers more advantages than the nitric acid solution.

The following assays of Lake copper made on 1 gramme of material closely agree with those of Mr. Heath, and may be of interest. The assays were made upon borings from the same ingot. The following results were obtained:

	A.	B.
Copper, per cent., ...	99.90	99.91

The mention by Mr. Heath of the fact that ammonia is formed during the deposition of the copper is important, and

should be taken into consideration, if other metals are to be determined in the solution. For instance, in the analysis of brass by depositing the copper in a solution containing considerable nitric acid, enough ammonia is often formed to prevent the complete precipitation of the zinc by sodium carbonate. I have repeatedly found results to come too low, if in such a case the ammonia is not removed before precipitation with sodium carbonate.

PROF. EDGAR KIDWELL, Houghton, Mich.: I would call attention to a method of charging storage-cells which, though rarely used, is particularly adapted to work of the kind here described. Owing to the large extent of ground covered, the alternating current is frequently used for incandescent lighting on mine-locations, while the outside lighting is done by arc-lamps. Because of the high first cost and trouble of looking after a rectifier to charge storage-cells from alternating-current mains, I believe it is in every way best, in such cases, to charge from the arc-circuit. The only reason why this is not more generally done is because of the element of danger existing when defective switching-apparatus is employed. Several years ago I had to handle a problem like this at the Michigan College of Mines, and designed automatic apparatus which has since been in continuous use and has given perfect satisfaction. The battery consists of over 60 cells. When charging, they are all arranged in series. Charging is done at night, when no one has occasion to use the battery. Beyond looking after the physical condition of the cells nothing needs to be done except to throw the main switch at any time during the day, if the cells are to be cut in or out that night. Even when charging with a 6.8-ampère current, sufficient energy can be stored during the night to supply the needs of the departments of physics and chemistry for the following day. Anyone interested can see the plant in full operation at the college.

The switching-apparatus is fully described in the *Electrical Engineer*, November 11 and December 16, 1896. They are free to anyone who cares to make them. The Automatic Circuit Breaker Company, Newaygo, Mich., made from my sketches the apparatus used at the college, and no doubt would be glad to supply anyone who needed similar apparatus.

FRANK KLEPETKO, Great Falls, Mont. (communication to the Secretary): Mr. Erwin S. Sperry mentions the fact that Hampe found from 0.007 to 0.02 per cent. of antimony in electrolytic copper. Mr. Hampe, at the time he made his experiment, may indeed have found this amount of antimony in electrolytically deposited copper, for the science of depositing copper electrolytically was then in its infancy. Such copper as was then produced by this method could not now be sold in competition with the metal deposited to-day in modern electrolytic plants.

At the Great Falls works we sample and analyze each shipment-lot of cathodes, also each shipment-lot of wire-bars. We invariably find that the total quantity of arsenic and antimony is below 0.005 per cent., and we do not think that this quantity of arsenic and antimony would have any deleterious effects on the qualities of brass. We find that this quantity of arsenic and antimony is not sufficient to prevent the copper from being used for high-conductivity wire. We do not find it necessary to estimate the arsenic separately from the antimony, separate determinations having shown that about two-thirds of the combined quantity is antimony and one-third arsenic. Our usual analysis, therefore, determines only the total of both.

As a further check on the chemist, we have a wire-drawing apparatus, and the wire is tested for conductivity. Our results check very closely with those obtained on samples of the same copper sent East for conductivity-determinations.

Mr. Sperry may have some information showing at what point or at what percentage antimony and arsenic as impurities of copper begin to affect injuriously the qualities of brass. I think the Institute would be pleased to hear from him on this point.

In this connection I give a description, written by Mr. G. A. Heberlein, our chemist, of our method of determination of combined arsenic and antimony in cathodes and wire-bars, and also for the separate determination of these elements, as practiced in our laboratory.

Determination of Arsenic and Antimony Separately in Wire-Bar and Cathode Copper.

Weigh out 100 grammes in portions of 25 grammes each. Dissolve each portion in 95 c.c. of concentrated nitric acid

(HNO_3) and 75 c.c. of water. Should the sample consist of borings, add the acid slowly. Treat each portion as follows: evaporate to one-third the original volume; take up with solution containing 800 c.c. of water, 22 c.c. of concentrated sulphuric acid (H_2SO_4), and 2.5 grammes of ammonium nitrate (NH_4, NO_3); electrolyze in a tall No. 6 Griffin beaker with a two-ampère current.

Use a platinum cylinder, 4 inches high and 3 inches in diameter, having 30 rectangular 1¼ by ⅛-inch perforations in two parallel rows ½ inch apart, for cathode. For anode, use a corkscrew spiral of No. 16 platinum wire, running at a distance of ⅛ inch parallel to the inside of the cylinder for the full length of the latter. The shape of the cylinder, and that of the spiral, insure a splendid circulation and a uniform deposit.

Fifteen hours should be sufficient for the deposition of 98 per cent. of the copper on the cylinder, and the deposition should not be carried further than this. Wash off spiral and cylinder. Combine the contents of the four beakers and evaporate till strong fumes of sulphuric acid indicate the absence of nitric acid. Allow the contents of the evaporating-dish to cool. After taking up with 300 c.c. of water, transfer to a No. 6 beaker. Wash out the dish with 20 c.c. of concentrated hydrochloric acid to remove any trace of antimony adhering to the inside of the dish. After combining the two solutions, dissolve 1 gramme of tartaric acid ($C_4H_6O_6$) in the warm solution. If any insoluble residue be present, filter off and treat according to directions given below.

Through the warm solution pass sulphuretted hydrogen till saturated. After 12 hours' standing, filter and digest the sulphides in warm sodium sulphide solution, 1.1 specific gravity, in a stoppered flask for five hours, which dissolves completely the sulphides of arsenic and antimony. Filter and precipitate in the filtrate the above-mentioned sulphides by the addition of dilute sulphuric acid. When the sulphides (mixed with sulphur) are settled, filter, and dissolve the contents of the filter in 1 part concentrated hydrochloric acid, 1 part water and chlorate of potash ($KClO_3$) sufficient for oxidation. Drive off the chlorine. Filter off the sulphur. Add hydrochloric acid until the filtrate contains two parts of hydrochloric acid (HCl) to one part of water. Add also 300 milligrammes of

tartaric acid. Pass through sulphuretted hydrogen to saturation. All the arsenic will be precipitated as a mixture of As_2S_3 and S, while antimony will remain in solution. Allow to stand 10 hours; treat the filtrate according to directions later given. Filter off the sulphides of arsenic, re-dissolve, and weigh ultimately as pyro-arseniate or as trisulphide.

Determination of Arsenic as Pyro-Arseniate.—Re-dissolve the sulphides in a solution containing hydrochloric acid, water and chlorate of potash. Drive off chlorine by heat. Add ammonia to alkaline reaction. Add an additional quantity of ammonia equal in volume to the volume of the solution. Cool, and add a few drops of magnesia mixture, containing one part maguesium sulphate, two parts ammonium chloride, eight parts water and four parts ammonia. Allow to stand 12 hours. The precipitate consists of magnesium-ammonium arseniate. Dry, heat gradually as prescribed by Fresenius, until the filter is consumed. Finish at bright red heat. Cool and weigh as pyro-arseniate of magnesia.

Determination of Arsenic and Antimony Together.

Instead of separating arsenic from antimony in the hydrochloric-tartaric acid solution which was finally obtained, dilute the solution to four times its volume; pass sulphuretted hydrogen to saturation; filter the combined sulphides on a 4 cm. filter (weighed and dried in weigh-bottle at 110° C.); wash with water, absolute alcohol, carbon bisulphide, and finally absolute alcohol again, very carefully; then dry at 110° C., cool and weigh.

By allowing to dried filters in weigh-bottles the same time for cooling in the desiccator when getting weight of bottle and filter-paper only, as afterwards when filtered sulphides are dry and ready for weighing, very uniform results are obtained.

Arsenic being precipitated as As_2S_3 and antimony as Sb_2S_5, their respective factors being 0.609 and 0.600, the factor 0.6 is used in calculating, from the weight of the combined sulphides, the percentage of arsenic plus antimony.

Determination of Arsenic as Trisulphide.—Re-dissolve the sulphides in a solution containing hydrochloric acid, water and chlorate of potash. Dilute to at least twice the volume. Pass through the warm solution sulphuretted hydrogen to satura-

tion, to insure the thorough reduction of the solution. Arsenic will be precipitated as trisulphide. After 5 hours' standing, filter. Wash carefully with water, absolute alcohol and carbon disulphide, in succession. Finish the washing with alcohol. Dry at 110° C. Weigh on filter as trisulphide.

Usually the trisulphide method is employed. For a check upon it, we use the pyro-arseniate method.

Treatment of Filtrate Containing Antimony.—Dilute the filtrate containing antimony to at least four times the original volume, and pass sulphuretted hydrogen through it to saturation. The antimonic sulphide (Sb_2S_5) is filtered off and converted into Sb_2O_4 by treating filter and sulphide in a weighed porcelain crucible with fuming nitric acid. Heat gradually to bright red heat. Cool and weigh as Sb_2O_4.

Treatment of Insoluble Residue from the Electrolyte.—Treat any residue from filtering by destroying the filter with fuming nitric acid. Fuse the residue with a mixture of sulphur and carbonate of soda. Dissolve in water and filter. Acidulate the filtrate with dilute sulphuric acid. Allow the precipitate to settle, then filter, and dissolve in hydrochloric acid, water and chlorate of potash ($KClO_3$). Add this solution to the filtrate from separation of antimony from arsenic.

This method of determining arsenic and antimony has been repeatedly checked by the sulphocyanide method, and found to be equally accurate and more convenient, as a great many determinations can be run through at the same time.

For very impure anode-copper, we use 25 grammes for determination, and obtain good results by cutting the electrolysis short when about 90 per cent. of the copper has been deposited on the platinum cylinder.

The copper should have the light rose-color and luster of copper deoxydized by hydrogen, and should show a very fine texture.

MR. HEATH: It seems that some statements in my original paper have started the ball of discussion rolling, so that the original topic has merged into a very interesting discussion by practical men on the highest amount of antimony and arsenic allowable in refined electrolytic copper, the influence of impurities on the conductivity of wire-bars and on the properties

of brass, and, finally, methods for the rapid estimation of these noxious elements.

Mr. Sperry has agitated an important question with reference to the limitations of the battery-assay by discussing the influence of antimony in solution, and emphasizing the fact that copper apparently deposits in a cleaner, brighter plate when a few drops of nitric acid are added to the sulphuric acid solution of copper.

It will be noted that my preceding paper recommends a carefully proportioned mixture of the two acids in preference to either alone; the proportion of sulphuric to be increased with the increase of impurities in the material analyzed.

There is some disagreement among experimenters concerning antimony.

Alex. Classen (on p. 93 of the book Mr. Sperry quotes) states that "Copper can be separated (from Sb and As) only when the quantity of arsenic or antimony is small, and when the current is allowed to act only long enough to reduce the copper."

Hampe, according to his article cited by Mr. Sperry, found a trace of antimony in electro-deposited copper. He used the current of six Meidinger cells (or probably three-tenths ampère, if the cells were in full working order), which may have been a little too strong, or too long continued after the deposition of copper was complete.

I have just finished the following experiment, which illustrates the point:

Experiment No. 8.

0.1 gramme of antimony was dissolved in 5 c.c. strong sulphuric acid, diluted to twice its bulk, and poured into a solution of 1 gramme of pure copper dissolved in 5 c.c. of strong nitric acid.

The solution was diluted and electrolyzed, without filtering off antimonic acid, with the use of a current which measured only 0.05 ampère with the Bunsen voltameter in circuit.

Under these conditions the plate remained perfectly bright until the liquid was colorless, and all but a trace of copper was deposited, when the plate quickly began to discolor, and the result was a little high. This corroborates Classen's statement.

I cannot agree with the inference that the battery-assay is not suited to any refined electrolytic copper, since I have, by Hampe's methods, proved much electrolytic copper to contain

only traces of arsenic or antimony, and have obtained as good rose-red plates of copper by the electrolysis of such material as with native metal.

In case a plate should be slightly darkened, it may be ignited to drive off a trace of impurity (as recommended by Classen); and the oxide may sometimes be removed from the deposit in a neat manner by placing the electrode in a beaker of distilled water, containing a few c.c. of acid, and reversing the current a short time, then reversing the current again. A positive plate, with considerable depositing-surface, is to be used.

I have been accustomed to remove antimony and arsenic from alloys, casting-metals, etc., in the usual manner, by appropriate treatment of the copper solution with hydrogen sulphide and alkaline sulphides, but this is a tedious process.

When arsenic only is present, Mr. A. H. Low's method of removal has appeared to be the most satisfactory.*

He prepares a solution of 2 grammes of sulphur in 10 c.c. of bromine, and adds 2 c.c. of this mixture to a hydrochloric acid solution of the copper, from which most of the free acid has been first removed by evaporation. The liquid is boiled one-half-minute; then 10 c.c. of strong sulphuric acid is added, and the flask is heated until the acid boils freely.

Mr. Low, however, says: "There appears to be no simple way to remove antimony."

Appreciating the value of a method to accomplish this, I have made a few experiments since receiving the foregoing discussion, and have found a method which, as far as present practice goes, proves very satisfactory for the removal of large quantities of antimony.

The process depends on the following statement of Watts:†

"Bromide of antimony ($SbBr_3$) is purified by distillation. It forms, on cooling, a mass of colorless needles, deliquescent, melting at 90° Cent., volatile at 270°. Water decomposes it, forming an oxy-bromide.

"This compound may be prepared by *distilling* a mixture of antimony sulphate and potassium bromide."

As arsenic would generally accompany antimony in copper, the tests were made as follows:

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* Peters's *Modern Copper Smelting*, ed. 1895, p. 52.

† Watts's *Dict. of Chemistry*, ed. of 1883, vol. i., p. 317.

Experiment No. 9.

0.1 gramme of metallic antimony was placed in a porcelain casserole (No. 3A; 3¾ inches, or 9.5 cms. in diameter) and dissolved in *aqua regia*, the nitric acid removed by evaporation and the solution treated with 10 c.c. of bromine containing 2 grammes of sulphur in solution.

The liquid was then evaporated on an asbestos plate, or sand-bath, to a syrupy condition; 20 c.c. of pure liquid bromine was then added, and the mixture carefully evaporated until the mass was pasty and the bromide (or sulphobromide?) of antimony volatilized slowly in white fumes.

Care was taken not to over-heat any portion of the dish; and in an hour or more nothing was left but a little carbonaceous film containing only *traces* of antimony.

Experiment No. 10.

0.1 gramme of antimony was dissolved in 5 c.c. of strong sulphuric acid and the bromine distillation conducted as in Experiment No. 9, until the sulphuric acid was also driven off. The removal of antimony was nearly complete, but not quite so perfect as in No. 9. Possibly, if the sulphuric acid solution had been neutralized with an excess of potassium bromide (Watts, *loc. cit.*), the action would have been complete.

Experiment No. 11.

A hydrochloric acid solution of 0.05 gramme of antimony was added to a hydrochloric acid solution of 1 gramme of pure copper, contained in a (No. 3A) casserole, and the solution was evaporated nearly to dryness.

It was then evaporated again, as in Experiment No. 9, with 2 grammes of sulphur dissolved in 10 c.c. of bromine until the mass was pasty. The syrupy residue was then treated with 20 c.c. of pure liquid bromine, and the evaporation was continued on the sand-bath until whitish fumes of antimony bromide were no longer evolved, and the copper salt was quite dry and light gray in color.

The casserole was kept covered a short time at first, to guard against loss by spattering.

The hydrochloric acid was, of course, partly removed, but it would possibly be better to use hydrobromic acid in the first

place, instead. The distillate on the under-side of the glass cover, which was at first employed, was dissolved and tested, but contained no copper. There ought not to be any loss of cuprous bromide at this temperature, if no part of the dish is too strongly overheated. Chloride of copper, if very strongly heated in contact with air, would be liable to loss.

The residue from Experiment No. 11 was found to contain only a trace of antimony; hence this method for the removal of the obnoxious elements, antimony and arsenic, has proved, so far, to be very satisfactory, and is recommended to chemists for trial. If it proves as successful with others as with the writer, it will enlarge the scope of the battery-assay, and make possible its direct application to the analysis of crude metal.

Methods have been published by well-known authorities for the subsequent estimation of traces of bismuth, if present in the deposited copper plates, or for the electrolysis under special conditions which will partially, at least, prevent the deposition of bismuth.

I will give a short account of other practical methods which I have successfully used for the estimation of arsenic and antimony in refined copper. It may be noted that bismuth—and tellurium—are more detrimental to the refined electrolytic copper than even arsenic and antimony. As bismuth has a greater tendency to deposit on the cathode than the latter metals, a trace of it at least is generally found in the refined metal.

The scheme of Mr. Heberlein, communicated by Mr. Klepetko, is evidently a very good commercial method. In a recent article* Mr. Herzig attributes a similar method to Mr. Thofehrn when in charge at the Anaconda works.

I have used Hampe's methods for the separation of traces of impurities from copper for years, and have been accustomed to separate arsenic and antimony from copper and from each other by one of two or three methods, according to circumstances.

Mr. Heberlein's plan of fractional precipitation of arsenic in very strong acid solution, in presence of antimony, must give a very good separation, except possibly (as stated by Mr. Herzig) when a little antimonic acid is present in the solution.

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* *Eng. and. Min. Jour.*, April 30, 1898, p. 518.

The employment by Mr. Heberlein of a large cylindrical platinum cathode with many perforations is a commendable plan for the rapid removal of copper from solution, and worthy of adoption by all laboratories.

After arsenic and antimony sulphides have, in the course of analysis, been separated from precipitated copper sulphide by extraction with pure sodium sulphide, and that solution acidified with dilute sulphuric acid and filtered, the residual sulphides of arsenic and antimony may be separated as follows:

1. By fractional precipitation; Mr. Heberlein's method.

2. By electrolysis;* Classen's method, modified by Lecrenier as follows:

The sulphides are dissolved in fuming nitric acid, the acid removed by evaporation, the residue dissolved in pure sodium mono-sulphide (60 c.c.), diluted to 200 c.c. after heating with sodium sulphite, as directed by Lecrenier, and the antimony removed by electrolysis.

One can obtain very good results in this way, but it requires care. A better method for the separation of arsenic from traces of antimony is the well-known precipitation of arsenic acid with magnesia-mixture. The usual method given by Fresenius gives low results, unless a correction is made for the solubility of the ammonium magnesium arsenate.

I have modified this process so that the loss is so small as to be practically negligible.

This is accomplished by keeping the total volume of the solution, after the arsenic is precipitated, at 15 c.c. only.

3. By a modified method described below. By this method the following results were obtained with quantities of pure arsenious oxide as large as would ordinarily be present in a refined electrolytic copper:

	Weight of arsenious oxide.		Gramme.
	I.	II.	III.
Pure arsenious oxide taken, . .	0.0118	0.0108	0.0013
" " " found, . .	0.0116	0.0110	0.00115
" " " difference,	−0.0002	+0.0002	−0.00015

Details.—After the purified sulphides of arsenic and a little antimony have been treated with fuming nitric acid or aqua-regia, and the sulphides and sulphur dissolved, the solution is

* Smith's *Electro-Chemical Analysis*, Phila., 1890, p. 88.

diluted somewhat and filtered from asbestos (if an asbestos filter had been used) and the clear solution evaporated to dryness on the water-bath.

Then dissolve in 1 c.c. of strong sulphuric acid 0.1 gramme of solid tartaric acid; add 5 c.c. of water, filter again (if not perfectly clear) through a very small filter into a No. 000 beaker; wash twice; neutralize with a slight excess of ammonia, and bring the solution either by dilution or evaporation, as necessary, to a total volume of 10 c.c., as shown by a marked paper pasted on the beaker. If much copper was dissolved in the first treatment with sodium sulphide, a second separation of soluble sulphides from that element may be necessary before this part of the operation.

Cool the solution (Vol. = 10 c.c.); add 1 c.c. magnesia-mixture (= 0.04 gramme $MgSO_4$) and make up to the 15 c.c. mark with concentrated ammonia.

The magnesia is made as follows (Fresenius's formula): By weight, 1 part magnesium sulphate, 2 parts ammonium chloride, 4 parts strong ammonia, and 8 parts water.

Filter after twelve hours; transfer ammonium magnesium arsenate to the 3 cm. filter with the aid of filtrate and wash finally with 10 c.c. of dilute ammonia (1 part to 3 of water).

Place the moist filter in a weighed porcelain crucible, moisten well with saturated solution of ammonium nitrate, ignite cautiously until the paper is charred, moisten with the same reagent again and ignite finally at red heat.

Separate the trace of antimony from acidulated solution as sulphide.

4. By the following method, when arsenic alone is present in copper. Much of the refined product on the market contains no antimony. To detect and accurately determine a slight trace of arsenic in such metal, I devised a very delicate modification of the old, well-known Marsh test, which has been improved (in accordance with the suggestion of Schmidt*) so as to yield strictly quantitative results in three hours from the time the residue of arsenious sulphide has been obtained by acidification of its solution in sodium sulphide by the usual method of separation from copper.

* *Jour. Anal. and App. Chem.*, vol. vi., p. 408, July, 1892, from *Zeit. Anorg. Chemie.*

This residue is attacked with fuming nitric acid, 5 c.c. of concentrated sulphuric acid added, and the nitric removed by heating. The remaining acid is diluted with 2 parts of water and the solution slowly passed through a Marsh evolution apparatus containing 25 grammes of powdered zinc and a drop of platinic chloride, and enough water to seal the tube. Then 0.5 gramme of pure tin foil is dissolved in 5 c.c. hydrochloric acid and added to the arsenic solution, before passing through the generator. This makes the evolution of arsenic as arseniuretted hydrogen strictly quantitative. The details of the method are fully described in the *Engineering and Mining Journal*,* with the proofs of its accuracy.

The evolution of the gas and the cooling and weighing of a section of the glass tube with metallic film do not require constant attention. The operation may be completed in two hours, and is very useful for the testing of copper containing no antimony, and only minute quantities of arsenic. It does away with any further purification of the first precipitate of sulphur and arsenious sulphide by reprecipitation and washing with various reagents on tared filters.

I hope to be able to communicate, in a short time, a perfected method, on the lines suggested in my original paper, for the direct electrolysis of copper containing both arsenic and antimony, thus widening the applications of the "battery assay" of copper.

The Influence of Lead on Rolled and Drawn Brass.

Discussion of the Paper of Mr. E. S. Sperry. (See p. 485.)

(Lake Superior Meeting, July, 1897.)

FRANK FIRMSTONE, Easton, Pa.: The effect of lead on brass seems to have been recognized by Berthier, and the facts published by him as early as 1818 (*Ann. des Mines*, 1st ser., iii., 1818, p. 347 *et seq.*).

He there says expressly that copper containing lead makes "dry" (*sec*) brass, and that, for the same reason, blast-furnace

* June 26, 1897, vol. lxiii., p. 663.

"cadmia" was preferred to calamine by the makers of brass by the cementation process.

In the same memoir he says that "dry" (*sec*) brass does not clog the tool, and that brass which does so is called "fat" (*gras*), is more ductile and tough, and is used for pin-wire.

Some Dike Features of the Gogebic Iron-Range.

Discussion of the Paper of Mr. C. M. Boss. (See p. 556.)

(Lake Superior Meeting, July, 1897.)

GEORGE H. ABEEL, Hurley, Wis. (communication to the Secretary): Doubtless Mr. Boss did not mean to be understood as saying that the dikes of this range are everywhere overlaid with iron-ore. I know of places where the dark, massive dike is found near the surface, but no ore is found on it. And his statement (p. 557) that "wherever the slightest inclination is shown it is always to the east," should be modified to "usually." There are several instances of a westward pitch, *e.g.*, at the Palms, Newport and Odanah properties, and also at the Montreal mine, where we encountered in the ore a small dike pitching west.

Mr. Boss's statement (p. 559) that the ore of the Pence mine lens "has all been removed from the Pence and Father Hennepin lands, and practically from the Montreal property, and is now being wrought by the Montreal Mining Company on the 'Sec. 33' property," does not convey the exact state of the latter company's operations. We are still mining ore on the Montreal property from the lens referred to, and expect to continue doing so for some little time to come. On the other hand, we are getting a shaft down to the ore on the "Sec. 33" property, but we have not yet encountered the dike. This can hardly be considered as working the ore. I do not understand upon what evidence Mr. Boss can consider the correctness of a previous hypothetical location of the dike to have been established "within 2 feet," when the dike has not yet been reached. Possibly he has been misled by hearsay reports.

The Efficiency of Built-Up Wooden Beams.

Discussion of the Paper of Prof. Edgar Kidwell. (See p. 732.)

(Lake Superior Meeting, July, 1897.)

PROF. HENRY S. JACOBY, Cornell University, Ithaca, N.Y. (communication to the Secretary): When a simple beam supports any given load, the lower fibers are in tension while the upper fibers are in compression, and if its cross-section is rectangular the neutral surface, where the stresses pass through zero from tension to compression, is at mid-depth, provided the elastic limit is not exceeded. In addition to the longitudinal stresses, there are horizontal shearing-stresses between the fibers which have their maximum value in the neutral surface, and which equal zero at the upper and lower surfaces, while there are also vertical shearing-stresses.

Let another beam of the same dimensions be placed on top of this one, and the load be doubled, it then produces stresses in each beam of the same character, magnitude and distribution as in the preceding case, provided there be no friction between the adjacent surfaces of the beams.

Next let these two beams be so firmly united that there will be no motion between them when the same load is again applied. All the fibers in the upper beam are now in compression, and those in the lower beam in tension, while the maximum stresses in the outer fibers are only one-half as large as before. The strength of the combined beams is therefore doubled. If, however, some motion occurs between the beams, then the stresses in some of the lower fibers of the upper beam pass from compression to tension, and those in some of the upper fibers of the lower beam from tension to compression, while the stresses in the outer fibers have a magnitude between the two values previously stated.

These fundamental facts relating to the flexure of beams indicate the necessity of carefully designing the keys or brace-

blocks of built-up or deepened beams in order to secure a high degree of efficiency. If the literature of the subject gives any indication of the general practice, it seems that but little attention has generally been paid to this feature.

On account of the compressibility of timber across the fibers, and its low compressive strength in this direction, as compared with the compressive strength in the direction of the fibers, it is practically impossible to secure the best efficiency with wooden keys whose fibers run across the beam. If the load be stationary, and it were possible to drive the keys to such an initial compression that no further deformation would occur after the load were placed in position, then the timbers might be prevented from sliding, and the built-up beam would have the same strength and deflection as a solid beam of the same material and dimensions. This would, however, be much more difficult with the arrangement of keys shown in Fig. 25 of Prof. Kidwell's paper than with the inclined keys in Fig. 13. Since the keys in either case tend to rotate, it would further be required that the keys be wedge-shaped in both directions. This, in turn, would considerably increase the secondary flexural stresses in the main timbers, caused by placing the bolts midway between the keys.

The deflection diagrams 7 and 8 indicate remarkably good results; but whether the same results may be secured with component sticks of say 12 by 12 inches will depend upon the practicability of driving the larger keys equally tight without crushing them in the operation.

Let beam No. 20 be taken as an example. If its deflection-curve in diagram 7 be produced to the axis, the initial deflection is about $\frac{1}{32}$ inch. This initial drop is due principally to the local adjustment of the bearing-surfaces, and weakens the beam, provided it deflects below the horizontal. If the beam be slightly cambered, so that after the application of a given load it does not deflect below the horizontal, it is reasonably certain that the fibers in the lower timber are all in tension. The diagram shows that under a load of 1400 pounds the deflection, less the initial drop, is $\frac{5}{32}$ inch or 0.156 inch. The computed deflection of a solid beam is 0.127 inch if the coefficient of elasticity E be taken as 1,000,000 pounds, and 0.141 inch if E be assumed as 900,000 pounds per square inch. The

difference between 0.156 and 0.141 is $\frac{1}{64}$ inch, or 10 per cent. of the former value. The corresponding stress in the outer fibers is computed to be 965 pounds per square inch. Since the coefficient of elasticity was not observed for the sticks used in the beam, it is not possible to determine what the efficiency is under safe unit-stresses. A part, if not the whole, of the difference in deflection might be due to the fact that the end-keys were placed 3 inches inside of the supports.

The efficiency, as determined by relative breaking-tests, is not useful for the purpose of designing structures whose stresses are never to pass beyond the elastic limit, unless the conditions are at least approximately the same in both cases, until rupture occurs. With the same load of 1400 pounds the deflection of beam No. 24 is $\frac{1}{64}$ inch greater than that of No. 20. With the same coefficient of elasticity its efficiency, based on deflection under safe loads, is less than that of No. 20, but, when based upon the ultimate load, it is found to be considerably greater.

In comparing beams with different kinds of keys it is important that both should be equally well proportioned. In beam No. 36, if the unit-stress in the outer fibers be taken at 1000 pounds per square inch, the corresponding load is 1045 pounds, and the horizontal shear on each side of the middle is 5940 pounds. For an allowable unit-stress of 1100 pounds on the ends of the fibers, the height of a key should be $(2 \times 990) \div (1100 \times 5) = 0.36$ or $\frac{3}{8}$ inch. The moment of rotation of the key is $990 \times 0.18 = 178$ pound-inches. Assuming that the pressure on the horizontal sides of the key varies from zero at the middle to 200 pounds per square inch (see table on page 808) at the ends, its length in the direction of the length of the beam should be at least 0.90 inch, or $2\frac{1}{2}$ times the height. As the keys in No. 36 are square, the beam probably yielded first by the keys crushing the sides of the fibers, and thus allowing considerable rotation, and hence also slip ping between the beams, thereby increasing the stresses in the outer fibers.

With the square keys the stress in each bolt is 1484 pounds, which, at 12,000 pounds per square inch, is just the strength of a half-inch bolt. A washer, $3\frac{1}{2}$ inches in diameter, is required if only 200 pounds per square inch be allowed. Unless the

bolts are drawn up to nearly the required safe tension before the load is applied, the compression of the timber under the washers will permit the timbers to separate, and thus allow the keys to rotate still more. This actually occurs when the beam is tested to destruction; and hence the strength of the deepened beam gradually approaches as a limit the condition of two superimposed separate beams, or an efficiency of 50 per cent., without, of course, reaching that limit.

The cast-iron keys in Fig. 29 were doubtless driven to such an extent that the grip of the ends of the fibers on each of the vertical sides of the keys prevented their rotation until the load became relatively great, and, as shown by the deflection-curves, no appreciable slipping occurred between the beams under safe loads after the local adjustment of parts when the loads were first applied. The yielding of the keys described above takes place in this case also, but within narrower limits, near rupture.

It may now be apparent why such poor results were secured with Brunel's beam. The keys are not properly proportioned and therefore develop an excessive moment of rotation and pressure on the sides of the fibers, and, being composed of wood, their compressibility increases it still further. Theoretically this form has the advantage, since the keys are not subject to shear directly, but mainly to compression. In order to avoid the compressibility of timber-keys, let them be made of cast-iron or cast-steel. To avoid the secondary flexural stresses, which are not inconsiderable, caused by placing the bolts between the keys, let a hole be made in the web of the cast-iron key so that a bolt may pass through it. The form of the key is preferably that of a web of uniform thickness united to a flange on its four sides. The connection between them should be well rounded on account of the well-known structural weakness of re-entrant angles in castings. The round hole in the center of the web should be several times the diameter of the bolt. The flanges bearing against the ends of the fibers must be proportioned to transmit to the web the pressure due to the horizontal shear, while the other flanges serve mainly to stiffen the web, although they necessarily transmit some compression. With this form the lever-arm of the resistance of the sides of the wood fibers to the rotation of the key does not appreciably

change when the key rotates slightly. The bearing area of these fibers may be increased by increasing the thickness of the flanges extending across the beam in order to reduce the length of the key if desired. In general, the key should be fully as long in the direction of the beam as its width. This makes the angle quite small between the resultant compression transmitted by the key and the axis of the beam.

As such a key or brace-block cannot be driven into place like a pair of wedges, the same result may be secured by somewhat reducing the distance of the seats of the brace-blocks in the lower timber from its center, so as to camber the beam when the bolts are drawn up. Two methods of doing this were given in an article by Prof. C. D. Jameson in the *Railroad and Engineering Journal*, January, 1890. The initial camber should be such as to allow for the deflection due to the local adjustment of the bearing-surfaces, and to prevent the beam from deflecting below the horizontal when supporting its safe load. The lack of camber in beams Nos. 39 to 47 contributes perhaps as largely to their greater deflection as well as lower ultimate strength as the proportion of the keys.

Since no form of deepened beam practically allows a continuous surface of the timber to resist the maximum horizontal shear, it seems to be impossible to make a deepened beam as narrow as a solid one of equal strength which contains a minimum amount of wood. The inclined key apparently deducts nothing from the shearing-surface, but on account of the reduction in thickness of the strip, which tends to shear between the keys, practical considerations make it advisable to deduct about one-half of the length of each key. Where square notches are cut into the timbers, as in Figs. 25, 26, 32, etc., to receive the keys, the timbers require relatively more width than in any other form, and hence contain more material. The end-keys should not be placed inside of the supports, because they cannot resist the horizontal shear produced beyond them.

It may be of interest to state briefly the conclusions drawn by Forchheimer from a series of experiments on deepened beams with wooden keys, made in Vienna in 1890.* They are: 1. The keys should have their fibers extending in the di-

* See *Zeitschrift des Vereines Deutscher Ingenieure*, 1892, pp. 100 and 629.

rection of the beam. 2. The keys should be twice as long as high, to prevent rotation. 3. If the timbers are separated by an air-space the keys should have an extension or neck between the timbers to prevent rotation. 4. The bolts should not pass through the keys, and thus still further weaken the section. 5. The compressibility of wood is so great that no reliance must be placed on the friction between the timbers on account of drawing up the bolts. This article also refers to some experiments on deepened beams in which the timbers are united by clamps driven into their sides.

Prof. Kidwell deserves much credit for his painstaking labor in constructing and testing so large a number of deepened beams of various types; and it is to be earnestly hoped that his investigation may lead to a more rational design of the simple structure known as a built-up or deepened beam. It seems almost a pity, however, that valuable time should have to be expended in exposing the defects of such a form as that of Clark's beam, which might be condemned on purely theoretic grounds; but if it shall effectually prevent the future employment of that form, even this part of the work will serve a useful purpose.

PROF. J. B. JOHNSON, Washington University, St. Louis, Mo. (communication to the Secretary): Prof. Kidwell merits the thanks of the engineering professions for his elaborate and conclusive experiments on the strength of built-up wooden beams. I do not see how the work could have been better done. The results are satisfactory both to the theoretical and the practical man, and will be regarded as authoritative and conclusive, so far as these forms are concerned. The fact is, for such large wooden beams, girders and joists as are used in practice, and when supporting distributed loads, their greatest weakness is in shearing or splitting longitudinally along their neutral planes. Any wooden beam, so loaded, having a length less than twenty times the height of the beam, is more likely to fail by shearing than by cross-breaking. This limit includes practically all lengths in actual use, and hence, we may say, wooden beams should be proportioned for longitudinal shear in place of cross-breaking. The necessity, therefore, of having great shearing resistance along the neutral plane becomes ap-

parent. If a solid beam will fail by splitting, much more will a composite beam, which is already parted here, show weakness from this cause.

The tests here reported only prove what theory has always shown, that it is practically impossible to put two sticks together, one above the other, in such a way as to equal the strength and stiffness of a solid beam of the same dimensions. The only way in which it could be done would be to glue them or cement them together, so as to make them in fact one solid stick. Whether or not this could be done in practice is doubtful. Perhaps plaster-of-Paris, after the surfaces had been painted or varnished, could be used, in connection with iron bolts to draw the parts together quickly, before the plaster hardens, and so cement the two sticks solidly as one beam. I am satisfied there is no system of keys, or joints, which can be made to accomplish this so long as this plane of junction remains open. Perhaps Prof. Kidwell will supplement his experiments with some tests of this sort, trying various cementing-materials. It might be that such a union could be made much cheaper than any system of keys, as well as far more efficient. At least, it would seem to be well worth trying.

B. E. Fernow, Division of Forestry, Department of Agriculture, Washington, D. C. (communication to the Secretary): Prof. Kidwell deserves special credit for subjecting with independent thought to exact investigation a problem of engineering with wood, which had so far not received any adequate treatment; the best authorities, like Rankine and Clark, contenting themselves with faulty theories, without experimental data to prove them, and, moreover, copying from each other without verification. His methodical procedure in carrying out a long and carefully designed series of tests, avoiding useless trials, his unusual circumspection in selecting and preparing test-material, and checking theories and results; and his recognition that workmanship is an essential element in the problem, are the more commendable, since wood-testing has, until lately, been carried on by engineers without such care and attention to fundamental details and without a proper conception of the difficulties involved in investigating this non-homogeneous material. Prof. Kidwell's paper must be acknowledged

as a decided advance in the knowledge of engineering with wood, and a first step towards unraveling one of the many unsolved problems confronting us in that department.

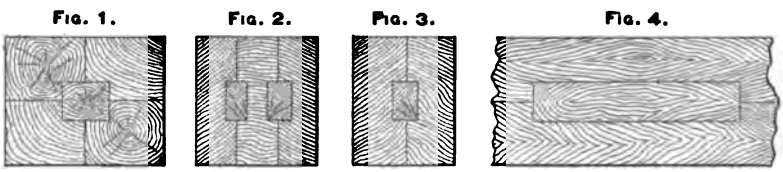
Nevertheless it is proper to point out that the subject has been by no means exhausted, and that the results stated in the paper must not be taken as final for built-beams in general, but only for the forms discussed. Moreover, in the comparison of solid with built-beams, Prof. Kidwell assumes that both contain wood of the same quality. He does not point out, and perhaps did not realize, that the great advantage of built-beams, which, with proper construction, might make them superior to solid beams, is the fact that for the parts of the former superior selection and disposition of material is possible. Solid beams are almost invariably full of defects which are invisible or unavoidable; they are never as well or as uniformly seasoned, and in shaping them, their fibers must be more or less cross-cut; while the planks for a built-beam can be selected free of defects, thoroughly seasoned, and cut and placed together with the most advantageous position of the grain.

The combination of various kinds of wood for various qualities (such as oak for tension-members) was perhaps not fully enough considered by Prof. Kidwell. Nor have the methods of combining component parts been exhausted. It would, for instance, by no means appear impracticable, from an economical point of view, to apply glue in the building of beams for interior constructions of a high order.

In short, notwithstanding his interesting experiments—or rather, without questioning their conclusiveness for the precise conditions and the special forms involved in them—I still venture to believe that by suitable selection and combination of materials a beam could be constructed which would be superior in all respects to a solid beam.

The value of Prof. Kidwell's work lies mainly, I think, in the proof that beams built in the manner indicated cannot be relied upon to develop much more or less than three-fourths the strength of a solid beam of equally good material, and that they also lose in stiffness; furthermore, in the bringing forward of formulas for designing built-beams, based on experiments, which will tend to do away with the arbitrary, unsubstantiated rules now in use.

I may add that there are several kinds of built beams in actual use which would appear superior to the forms tested by Mr. Kidwell. In one of these, which requires little skill in workmanship, in which the keys are not liable to displacement and a full measure of stiffness is secured, the keys consist of square compression-pieces, inserted longitudinally by squaring out the necessary space of the component parts as indicated in Figs. 1 to 4, of which Figs. 1, 2 and 3 represent cross-sections through the keyed part, and Fig. 4 a longitudinal section. The components are bolted together. It is necessary to make the



Sections of Built Beam.

keys sufficiently long to avoid shearing. A combination of three components would appear most desirable in order to have the median line in a solid part of the beam.

William Kent, New York City: Tredgold's *Principles of Carpentry*, a late edition of which is cited by Prof. Kidwell in his bibliographic list, was first published in 1820, and the author died in 1829. How much of his original work has been modified by later editors in successive editions I do not undertake to say, but it is highly probable that engineers in 1898 are still employing as a basis of their calculations, to a considerable extent, observations and formulas promulgated by Tredgold nearly eighty years ago. However great his merit and authority as a pioneer, the improvement in methods and the enlargement in scale of experiments have made it unquestionably necessary to revise fundamental data of so ancient a date.

I may mention, in illustration of this proposition, an experience in a cognate department of engineering, in the course of which I had occasion to examine the fourth edition of Tredgold's work on cast-iron, edited by Hodgkinson, to find out what Hodgkinson's experiments were on which Gordon bases his formula for proportioning cast-iron columns, which formula is now almost universally used. I found that Hodgkinson's experiments on flat-ended columns were made on pipes not over

0.35-inch thick, and not over 2¼ inches in diameter. We all know that no conclusion can be drawn as to the strength of a piece of iron 1 inch thick from the test of a piece that is cast only ¼ of an inch thick. But Hodgkinson made those experiments, and the whole world since that time has been using formulas based upon them. The building-law of New York City requires that we shall use Gordon's formula with a given co-efficient, and then says we shall use a factor of safety of 5. The New York Building Department, after the Ireland building collapsed two years ago, had some of the columns tested at Phœnixville. The reports of these tests were published a few weeks ago (see *Engineering News*, Jan. 13 and 20, 1898), and show that instead of these columns having a factor of safety of 5, which the building-law says they must have, they had a factor of safety of somewhere between 2 and 2.5. There have been three or four buildings in the last four or five years that have fallen down in New York, and the probable cause of the collapse of those buildings was the cast-iron columns; yet the New York building-law to-day compels inspectors to pass columns built on that formula based on these inadequate experiments of Hodgkinson, made sixty years ago. It is high time we revised the formula based on these old experiments.

HAROLD D. SMITH, Taunton, England (communication to the Secretary): Prof. Kidwell's paper is a most interesting account of a very complete series of tests, which were made with great care to escape all avoidable errors.

The method of cutting the components of each beam from the same plank diagonally is most ingenious, and eliminates as far as possible the serious risk of error due to the heterogeneous structure of pine timber.

The fact mentioned (page 736) that "in nearly all of the tests the tension-side failed first" is the exact converse to what took place during my own experiments. In nearly every case in these the upper portion of the beam showed before fracture a crumpling outwards of the fibers, which is characteristic of compressive failure of timber.

The fact that in the direct tests the tensile strength came out much higher than the compressive strength, gives striking confirmation to this observed process of the failure.

I cannot agree with the views given on pages 749 and 750 of the paper with regard to the inclined bolts advocated by Rankine and used by Brunel, for an inclined bolt must be stretched before the beam can give, and so helps the beam. Relatively to the bottom member, the top member moves outward from the center of the span if free. If constrained by bolts or keys, its effort results in compression, the reaction being taken up as tension by the lower member. Hence the bolts tend to become less inclined to the neutral axis.

The internal stresses in the plane of neutral axis in a solid beam are, as Rankine pointed out, shearing-stresses of equal intensity in the plane of the neutral axis and normal to it, and he showed how these could be resolved into equal tensile and compressive stresses at an angle of 45° to the neutral axis.

Suppose that coincident to the neutral axis there is a plane through which shearing-force parallel to it cannot be transmitted. This would be represented by a beam of two pieces, with frictionless rollers substituted for the keys. In this case the forces acting across the joint would be wholly normal to it, and the two pieces would act as separate beams, with the exception that the top-piece would distribute the load more or less over the bottom one.

It would appear that a built-up beam must occupy a place intermediate to these extremes of a wholly normal compression, and a compression and tension inclined at 45°, and that therefore the bolts should have an inclination of something between 45° and the normal.

This view was confirmed by my experiments, in which the bolts tended to take up positions less inclined to the neutral axis.

Fig. 13, on page 752 of the paper, hardly represents Brunel's beam fairly, for he was careful to have the bolts as near as possible to the keys, so as to counteract their tendency to revolve and open the joint. In this he was so successful that in the two I tested it was impossible to force a thin-bladed knife into the joint between the pieces.

The sketch* in my paper is not as clear as it should be, but

* *Minutes of Proceedings of the Institution of Civil Engineers*, vol. cxxviii., part 2, page 332, Fig. 8.

in the text it is stated that the beams were full-size models of a continuous-beam viaduct. The end-bolt was inclined in a direction opposite to the others because it formed the first bolt in the next span in the actual viaduct.

PROF. KIDWELL: I feel highly gratified at the interest taken in this work, and at the fact that the discussions have been contributed by such able authorities. My warmest thanks are extended to them and to many others who have written to me and given valuable suggestions and information.

I purpose to continue the experiments if circumstances permit. I will therefore reserve for a later paper a full consideration of all the points brought out in the discussions; some of them, indeed, cannot be answered at all until further work is done. I shall therefore refer briefly to a few matters which can be profitably taken up at present.

It is true that much attention was given to Clark's beam. There is good reason for this. That construction has been much used, and is by many considered good. Extremely bad results which I noted in one case where it was used led me to make the experiments necessary to prove that this construction had better be let alone. Later on, I thought it well to extend the investigation to other constructions as a matter of scientific interest; hence Clark's beam, and the defects in it, are responsible for the existence of this paper.

I had difficulty in getting the timber necessary for the tests. My supply was limited; every piece of it was used up; and I had to limit the experiments accordingly. This readily explains why many things which evidently ought to have been done were omitted for the present.

That the efficiency based on the breaking-loads is not the most scientific mode of judging the value of a beam is quite true, yet I employed it because that method is the one current among engineers. In further work I shall also attempt to obtain efficiencies based on the coefficient of elasticity, in line with Prof. Jacoby's valuable suggestion.

The majority of my correspondents think the fibers in the wooden keys should run in the same direction as those of the beam, not *across* the beam, as I had them. With this disposition of the keys, the evils due to shrinkage will be lessened;

the indentation of the keys will also be decreased, and hence the slip in the joint will be less. Further experiments are needed on this point.

A correspondent also writes me: "There is a prejudice, whether well founded or not, against using keys of either of the woods mentioned in contact with white pine timber or its kindred." I believe this prejudice has some foundation in cases where the beam is used for outside work. For beams in buildings it may or may not be true. I have no data on the subject, and will thank anyone who will write me as to his experience in this matter.

Prof. Jacoby is quite right in his statement that the wrought-iron keys were not properly proportioned. This was not an oversight, however. Mines can seldom obtain castings quickly enough to use in emergencies where a large beam might be wanted; so I made the wrought keys of material that would be ready at hand, and drove them with little regard to proportion or spacing, as I wanted to see how they would act. The results are surprising to me, as they have been to others, and are no doubt due to the firm bite the keys took on the end-wood in the slots.

Prof. Jacoby, Mr. Smith, and several able correspondents think that Brunel's beam is a far better construction than one would judge from my experiments and conclusions on it. I regret that lack of material prevented more experimentation on this beam. While it is true that my keys were not nearly so long as they might have been, it must be noted that increasing the length of key cuts away more of the solid material of the beam. Again, I do not think it good policy to put a bolt through a key, as it makes the net section of the beam too small. A bolt on each side of the key is better, and this is the plan I shall adopt in future tests on this beam. At present, I am not partial to inclined keys, and will therefore take especial care to experiment thoroughly and think the whole matter out.

Joints made with glue and cements were considered, but had to be omitted, owing to lack of material and time. Two very serious questions arise concerning such joints—questions, the answer to which must be gathered from long use under actual conditions rather than from laboratory experiments.

Would not any cement like plaster-of-Paris gradually disintegrate under the vibration which necessarily occurs in every floor? Destruction of masonry-joints through vibration of machinery in buildings is well known. The Bureau of Engraving and Printing had to be removed from the United States Treasury Building at Washington on this very account.

Can we really trust glue in such important work as a built beam? I would hesitate to do it. It is questionable whether, with such large timbers, the truth of surface and heavy pressure essential for a good glue-joint could be got with reasonable cost. Would the joint hold for all time, even if good at the start? One who has seen how apparatus-cases, etc., go to pieces after a few months' service will have doubts on this point. Again, during the erection of most structures the beams are exposed to weather which would seriously impair glue-joints. Other troubles must be considered, such as the effect of a burst water- or steam-pipe, etc. I believe, however, that dipping the wooden keys in glue before driving would be a good thing.

In regard to Figs. 2 and 3 in Dr. Fernow's discussion, it should be said that this investigation was confined exclusively to deepened beams. Forms made by placing side by side a number of full-depth narrow strips, then securing the whole, were purposely excluded, and will be taken up as a separate problem.

I have been fully alive to the fact that a built-up beam is advantageous because it offers the opportunity to make a superior selection and disposition of material; and I have used them myself on this account. Yet I fear that, considering the question of cost and difficulty of getting workmen who have the requisite skill and comprehension of the importance of the problem, we shall fail to verify, commercially, at least, Dr. Fernow's prediction that "by suitable selection and combination of materials a beam could be constructed which would be superior in all respects to a solid beam." I hope I may prove to be wrong in this view, yet I think the outcome will show that I am right.

While the amount of work done thus far is by no means small, I am quite aware of the fact that the subject has hardly

been more than scratched into, and that a vast deal of work remains to be done before we can feel that all the essential points have been worked out. This work I hope to do if I can obtain the necessary mechanical facilities.

The Genesis of Certain Auriferous Lodes.

Communications in Discussion of the Paper of Dr. John R. Don. (See p. 564.)

(Atlantic City Meeting, February, 1898.)*

JOSEPH LECONTE, Berkeley, Cal.: I have read with some care and with extreme interest the work of Dr. Don, and have no hesitancy in expressing my high estimate of its value. We have here an example of laborious work undertaken in the true scientific spirit and by right methods. Loose statements and rash conjectures are here brought to the test of chemical analyses. By such work only may we hope to reach reliable conclusions and finally to solve the complex problems presented by the occurrence of ore-deposits.

Such work as this is not only scientific but is in the highest degree practical; for while a crude and imperfect science, by interfering with the results of approved empirical methods, may be positively hurtful, a more perfect science must eventually become the only sure guide to practice. Such a more perfect science can only be reached by such work as Dr. Don's.

I am sure every member of the Institute will unite with me in the hope that Dr. Don will continue his work, and that his example will incite others to similar work.

S. F. EMMONS, Washington, D. C.: I desire to bear my humble testimony to the great value of Dr. Don's paper to the science of ore-deposits, the thoroughness and accuracy of his work, and the immense amount of care-taking and tedious labor which it represents. The only regret with regard to it is that it could not have been published in full. No more important paper in its line has ever appeared in the *Transactions*. Indeed,

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* See Preface.

this is a line in which far too little has been done anywhere. Geologists are not often sufficiently trained chemists to carry on such work, if they had the time; and for mining geologists in our country the press of work in other directions is so great that they could not give the necessary time involved in this class of work, if they were so inclined. The chemist, on the other hand, is rarely enough of a geologist, or so placed, as to get sufficient field experience to keep thoroughly in touch with the processes of nature as shown in mine-workings. By an organization like the U. S. Geological Survey, where the chemist and the geologist might work in harmony for a common purpose, it would seem that investigations into terrestrial chemistry might best be carried on; and fifteen years ago, in connection with Mr. Hillebrand, one of the most thorough inorganic chemists of the day, I had planned such a line of experimental work, which I hoped might be continued as part of the regular chemical work of the Survey. But the powers that were willed it otherwise. My colleague, Mr. Becker, has made some important researches in this line, especially with regard to the natural solvents of gold which do not appear to have come under Dr. Don's notice. Such work is necessarily very slow, and Dr. Don's paper, as he tells us, covers the results of seven years' labor.

I do not propose to discuss Dr. Don's paper from a chemical point of view, but only to consider the deductions that may be made from it from the geological side; and I do this with some hesitation, because, not being able to read his statements in detail, I may not infer correctly what his actual conclusions were. I must say, however, that he seems to be most broad-minded in this respect, and his work presents a pleasing contrast to parts of the great paper of the lamented Posepny, where the effects of the latter's recent contest with Sandberger seemed to make him look at Nature through *ascensionistically* colored glasses.

Dr. Don's first and most important conclusion from his tests is that gold does not occur in the rocks of the regions investigated by him as an original constituent of the bisilicates, and that where it is found in these rocks it is associated with sulphides, mainly of iron. His inference seems to be that it cannot be original in the rock, because pyrite is necessarily a sec-

ondary constituent, that is, one introduced after the rock consolidated. Now, my work for the past ten years has been bringing me more and more to doubt the adequacy of the bisilicates of eruptive rocks as a source of the metals for our ore-deposits, and, especially in the case of gold, to look to the pyrite, if not as a source, as the visible accompaniment. Where there is no pyrite in the neighboring eruptive rocks, I have not found that the veins are usually rich. But I have not, therefore, abandoned my belief that eruptive rocks, similar to those we see at the surface, are the source of supply from which the great majority of our ore-deposits have been concentrated. Pyrite is not necessarily a secondary constituent in such rocks, as seems to be tacitly assumed by many. On the contrary, most petrographers admit the existence of primary pyrite, though they do not generally appreciate its importance in the study of ore-deposits. Lindgren has lately discussed primary pyrites as a product of magmatic consolidation in the gold-bearing rocks* of California. I cannot even feel absolutely sure, in spite of the apparent conclusiveness of Dr. Don's investigations for his regions, that the bisilicates of our Rocky Mountain eruptives may not contain some of the metals, but must wait until similarly exhaustive tests have been carried on here. Lead and cobalt have been found by Mr. Hillebrand in the bisilicates of some of the eruptive rocks of the Ten-mile district, and lead and silver similarly by Mr. Eakins in the bisilicates of granite at Silver Cliff, Colorado.

It seems important to note that the lateral secretion theory which Dr. Don's tests seem to disprove is not the one that has been generally advocated in the United States; for I fancy few American geologists believe in the narrower view advocated by Sandberger, that the metals are derived necessarily from the immediately adjacent country- or wall-rock. Dr. Don says that he believes the gold of New Zealand is derived from rocks deeper than any now exposed at the surface there, but adds that he "is not concerned with the question whether this source is the vague barysphere, with its somewhat apocryphal contents of heavy metals." While, therefore, no longer a be-

* "Gold Quartz Veins of Nevada City and Grass Valley," by W. Lindgren, *U. S. Geol. Survey, 17th Ann. Report*, Part II., p. 94.

liever in Sandberger, he is apparently not willing to subscribe to the extreme views of Posepny. It has been the fashion for some time to decry Sandberger; but I think his work has been of the utmost value to the study of ore-deposits, even if his ultimate conclusions are not admitted; for he aroused us from the unthinking belief that the metals necessarily came from unknown depths, which it was fruitless to speculate about or to try to investigate. He started in movement the pendulum of thought, which had so long been stationary at that point; for a while it swung on his side, then back again to the ascensionist side. Now, with some geologists, it has taken a very strong impetus in the direction of the long-ago abandoned sublimation-theory, or something very like it. The Swedish geologists represented by Vogt, a disciple of the great petrographer Brögger, consider certain workable masses of iron and other ores as segregations from the still molten mass, or direct products of magmatic differentiation. Other minerals occurring in pegmatite veins, and some metallic oxides, like tin, for instance, they consider to have been deposited in the last stages of consolidation of a molten magma by a mixed process, which they call pneumatolysis, and which brings in the agency of water expelled during cooling from the igneous magma. This comes back to the old French theory of aqueo-igneous fusion in the presence of certain substances, such as fluorine, boron, chlorine, etc., which were called *agents minéralisateurs;* but it is less purely theoretical, in that it is founded on certain facts of observation in Nature.

Vogt is at present preparing a new book on ore-deposits, in which he will doubtless extend very widely the scope of his pneumatolitic processes. But the applicability of either the French or Swedish theory must be limited by a very simple geological condition, which is—whether the fissures in which ore-deposition has taken place could have been formed during the final consolidation of the eruptive magma. In most of the important ore-deposits which I have had opportunities of studying, the fissures or fractures which have served as ore-channels were the result of earth-movements that took place long after the entire consolidation of the eruptive magmas. In some cases, even, there is evidence of several such movements before the ore-deposition. My belief, as I have had occasion to state

already, is that, by reason of some process which, for want of a better name, we may call magmatic differentiation, certain portions of an eruptive mass are richer in metals than the average; and that from such portions circulating solutions have abstracted these diffused metals and deposited them in a more concentrated form in favorable situations in rocks in the vicinity. The original bringing up of the metals from the bathysphere was, however, accomplished by the eruptive magma before consolidation. Present ore-deposits are in this sense the result of a secondary concentration at only moderate depths.

Dr. Don's investigation of the processes going on, or that may go on, in the vadose region affords interesting and useful data. He first assumes that deposits are generally richer in the precious metals above than below the water-line—a perfectly justifiable assumption, in my opinion—and then presents alternative explanations, one based on the ascension, the other on the lateral-secretion theory; but he does not mention the most obvious explanation of this fact, namely, that the baser metals, which are usually in far greater amount than the precious metals, form by oxidation more readily soluble compounds, and are therefore to a much larger extent removed in the vadose or oxidizing region, while the precious metals are for the most part either not dissolved or are re-precipitated. Hence, the specific gravity of the whole mass of vein-material is decreased in the vadose region; and a given bulk, though it may have absolutely no more of the precious metal in it than before oxidation, contains a relatively higher percentage, by weight, of the precious metals. There may also be in places an actual enrichment, due to the ready precipitability of the precious metals. I have myself seen cases where, in a zone immediately above the water-line, the ore-body was not only very much richer than in the unoxidized portion below, but also richer than the average of the oxidized ore above, showing that the precious metal had been in a measure leached down and re-precipitated. Dr. Don himself shows various ways in which gold might be dissolved in the vadose region and re-precipitated lower down; he also thinks it may have been carried down mechanically; the result of his investigations of mine-waters being that, where they are found to contain gold, it is usually in mechanical suspension rather than in solution.

One of the most interesting parts of Dr. Don's investigations is that relating to the gold contents of sea-water, in which he has not contented himself with the vague statement that the metal does occur, but has made a quantitative estimate of its amount, which must be translated into the metric system in order to become intelligible to any but English chemists. The average content as determined by him is 0.0071 grain to the ton, which means, if I have calculated correctly, something like twelve million dollars' worth to the cubic mile—a large sum, if it could be gotten out. But Dr. Don has shown by a pretty exhaustive series of experiments that there is no probability that any natural precipitant would throw down any measurable portion of it.

GEORGE F. BECKER, Washington, D. C.: Dr. Don's paper is an extremely important contribution to mining geology, and worth the vast amount of labor which it has manifestly cost. I trust that the paper in the *Transactions* will prove to be only a preliminary abstract, and that the entire memoir will soon appear.

The greater part of the conclusions which Dr. Don reaches are quite in line with those to which Mr. Lindgren and I have been led by studies of the gold deposits in this country.* I have been greatly impressed of late years with the character of the auriferous wall-rocks of gold-quartz veins. It seems to be true in most cases for deep mines, that the gold in the walls is contained in the sulphurets; that it is of smaller fineness than the vein-gold; that the sulphurets are pyrites; and that gold and sulphurets diminish rapidly in quantity as the distance from the vein increases. Sometimes the wall-rock pyrite is almost barren. There appear to me to be two sorts of wall-pyrite. One seems to be the result of the action, on ferro-magnesian silicates, of the sulphides of hydrogen or of the alkalies, as I pointed out in treating of the Comstock lode. This variety is sometimes, and I suspect usually, almost worthless. There are other impregnations of pyrite which seem to me to have permeated from the vein-fissures, in solution, as double sulphides of the alkalies and iron. I have studied such solutions in a

* Parts of these studies have been printed in the *U. S. Geological Survey, Ann. Reports*, 14th to 18th.

memoir on quicksilver-deposits, and shown that solutions of alkaline carbonates, partially charged with hydrogen sulphide, will take up notable quantities of many sulphurets, and also of gold. Dr. Don has, I think, overlooked this investigation. Mr. Dölter has since succeeded, where I failed, in dissolving lead and silver sulphides in a similar menstruum.

Wall-rocks appear to be much more permeable by solutions of some substances than by those of others. Native gold, quartz, and all the sulphurets excepting pyrite, are for the most part retained in the veins, while carbonates seem to penetrate freely into the wall-rock. This indicates an osmotic separation of the metal-bearing solution, as I pointed out years ago.

That some undecomposed rocks (granite, andesite) contain free gold in clearly visible particles appears to be quite certain. There is also strong evidence for the strange hypothesis that some sulphurets in massive rocks are original constituents. I entertain no doubt that some eruptive rocks associated with veins are truly metalliferous. Nevertheless, the possibility presents itself that in such cases vein and rock each derived its metallic contents from a common source, miles beneath the surface. Such an hypothesis is not without difficulties of its own, but accords better with the osmotic phenomena than the theory of derivation from wall-rock.

Dr. Don's remarks on secondary re-crystallization are interesting. That sulphurets are regenerated is certain. The question of the re-crystallization of gold needs more research.

ARTHUR WINSLOW, Kansas City, Mo.: Dr. Don's work, even as presented in the abridged form of his Institute paper, is a notable example of painstaking, conscientious research, for which the author deserves great credit. The field and literary work alone must have been very great; but, in addition, the investigation involved many special experiments on a large scale, and the testing and assaying of over 400 bulky samples, with more than 100 other assays and analyses, which, to any one familiar with such tasks, seems a stupendous labor. The results are especially of quantitative value, furnishing many exact facts and definite data which will control speculation, and upon which theories can be built up. Therefore, in whatever estimation one may hold Dr. Don's methods and results, or

however one may differ in the conclusions to be drawn, all, I think, will agree in admiring the courage and entire disregard for labor which prompted and sustained his inquiries.

The investigation is, first and principally, one to determine whether the ores of certain auriferous lodes of New Zealand, Victoria and Queensland were formed by lateral secretion or by ascension. Secondarily and incidentally, it is a valuable contribution to the general subject of the genesis of ore-deposits.

The work is of special interest when considered in connection with the hypothesis advanced by Mr. T. A. Rickard* for the origin of the ores of the Bendigo reefs of Australia. Indeed, it would appear as if Dr. Don must have had this hypothesis immediately in mind in planning his work, so directly does he meet the questions which arise in reading Mr. Rickard's paper. Passing by his experiments and conclusions as to the precipitation of gold in marine sediments, and also allied inquiries of somewhat negative value, the results of most positive and emphasized importance are his analyses of the country-rocks for the detection of gold. These are principally interesting in the uniformity with which they show that the metal was not detected, or was found to be present only in very minute quantities, in the normal country-rock; also that, where its presence was detected, it was associated with sulphides, and that both the quantity of the sulphides and the richness in gold increased as the lode or other form of deposit was approached.

The question which arises, and which to me seems of great significance, is as to the sufficiency of the evidence. Dr. Don's assay-results marked "Nil" as to gold-contents must be considered as relative. They mean simply that no gold was detected by the method he used. It is probable that gold and other metals are generally diffused through the rocks of the earth's crust, and are represented normally by amounts proportional to their abundance and the solubilities of their naturally-formed compounds. The researches of Forchhammer, Dieulafait, Bischof, Sandberger, Malaguti, and others, referred to in my report on the lead- and zinc-deposits of Missouri,† show the frequent diffusion of copper, lead and zinc in rocks,

* *Trans.*, xxii., 289.

† *Reports Mo. Geological Survey*, vol. vi., 1894, pp. 30 *et seq.*

minerals and waters. Mr. Robertson's analyses of crystalline and clastic rocks of Missouri, in connection with the same report,* sustain this belief. Proof that such diffusion is general would seem purely a question of the amount of rock used and of the refinement of method. The refinement of Dr. Don's methods may be inferred from the fact that one-tenth of a grain per ton is the lowest result given in figures. There is nothing between this and the "Nil" results. Gold to the amount of more than one-tenth of a grain he shows to be frequently presented in the country-rock, remote from lodes, and even amounts of over one grain were found. On the assumption that the gold has been derived from these rocks, such gold now found to be present must be a residuum after the process of leaching. The question arises, what gold-contents are required in a country-rock that it may be regarded as a source of the ores in the veins traversing it. Assuming one-tenth of a grain to the ton, it would take just 4800 tons of rock to supply the gold for one ton of one-ounce ore, if all the gold were removed from the country-rock into the veins. On this basis one cubic mile of rock would supply 2,300,000 tons of such ore. Mr. Rickard's hypothesis, which ascribes the gold to sea-water retained in the sediments, would hardly call for a greater original concentration, seeing that he allows only one-thirteenth of a grain per ton to the sea-water of the present day.

The conception of lateral secretion as a process whereby the contents of the ore-body simply trickle in from the sides or walls can, of course, not be sustained, even though the name may seem to justify it. To harmonize with the ideas now more generally held, the flow and transmission of solutions must have been in devious directions, according to hydrostatic and thermal conditions, and according to the structure of the rocks. The solutions would naturally follow fissures, and flow into open spaces, either in ascending or descending currents. They would at the same time penetrate the wall-rocks of such openings, and when they contained gold these wall-rocks would be enriched. Thus, an increase of the mineralization, or of the gold-contents as the vein is approached, is not incompatible with the large conception of lateral secretion.

* *Reports Mo. Geological Survey*, vol. vii., pp. 479 *et seq.*

It is true that such an expansion of the original idea of lateral secretion appears to rob it of its distinctive features, so that it seems in fact to embrace the rival process of ascension, and to leave no room for the separate existence of the latter. This is not, however, the case. The essential idea of the ascension theory is that metals and other constituents of lodes are derived from a deep-seated source of supply—a sort of treasure-house of nature—whence they were transported for the use of man by the agency of ascending solutions or vapors which followed the paths of profound fissures. The opposed idea of lateral secretion, as developed, if not precisely so expressed in recent writings, is that these constituents of lodes are distributed and diffused in greater or less quantities throughout the rocks of the earth's crust—in such rocks as are or have been exposed at the surface—that they have been gathered from these rocks and concentrated in ore-deposits by the action of percolating waters of various compositions, assisted by chemical, hydrostatic, thermal and structural conditions, such as are brought about by sedimentation, by earth-movements, by injection of volcanic rocks, etc. In cases of the more abundant metals, such as iron and manganese, this process is plainly recognized, and the requisite conditions are commonly presented. In the case of less common metals, such as copper, lead and zinc, the process is not so simple or so generally to be observed. In the case of the precious metals, which are rare and difficult of solution and transportation, more powerful factors and special or stronger reagents, such as highly-heated solutions, seem necessary to produce the result. Thus, according to the lateral secretion theory, our own home country-rocks are our treasure-houses, and we are not dependent upon the foreign depths of the earth for our metals. Such may be in some cases the source of supply, but the aggressive secretionist is disposed to throw the burden of the proof upon the ascensionist, and to maintain that his rule is the more satisfactorily proved by the removal of successive supposed exceptions.

W. P. Blake, Tucson, Arizona: As appropriate to the discussion on the origin of quartz-lodes and the deposition of gold, I desire to direct attention to the very general association in

California, especially in the Jura-Trias and Carboniferous slates, of gold with slates containing a large amount of carbonaceous matter. This association is particularly noticeable in the Princeton mine, on the Mariposas estate; in the Keystone mine, Amador county; in the Plumas mine, and in many other mines in the main gold-belt of the State. It has been noted also by Mr. Ross E. Browne, who, in a recent article, gives three cross-sections showing the mother-lode traversing soft black slate. He describes the lode as a "continuous series of parallel quartz-veins, following more or less persistently a narrow belt of soft black slate."*

At the Arizona School of Mines a sample of coal has been received from Wyoming, which is said to contain gold. Preliminary assays, made with great care, show an appreciable amount of the metal; but further investigation of the source of the coal and assays upon other samples are desirable before attaching special value to this form of association of gold and carbon.

The Accumulation of Amalgam on Copper Plates.

Continued Discussion of the Paper of Mr. R. T. Bayliss. (See vol. xxvi., pp. 33 and 1039.)

(Atlantic City Meeting, February, 1898.)†

C. N. FENNER, New York City (communication to the Secretary): A possible explanation of some of the phenomena of amalgamation cited by Mr. Bayliss and others has occurred to me.

We know that if, to a quantity of mercury, gold and silver are added until a putty-like paste is formed, and if this mass is spread on a flat surface and pressure is applied, a more fluid portion of the mercury will be squeezed out from the rest, carrying gold and silver with it. The proportion of gold to silver will depend, I think, upon various causes and conditions—the relative chemical affinity which gold and silver have for mercury; the amount of pressure used; the presence of other metals;

* *Min. and Sc. Press*, San Francisco, Jan. 29, 1898, p. 105.

† See Preface.

possibly also the temperature and other minor influences. In most cases, however, the proportions of gold and silver in the two portions will be quite different. This is in conformity with the generally accepted theory that amalgams form definite chemical compounds, which, however, are often either dissolved in an excess of mercury or mixed with an excess of the solid metal.

This operation must not, of course, be confounded with the process of straining the mercury through chamois, which latter holds back nearly all of the foreign ingredients and permits the mercury to pass through.

In order to confirm and illustrate the point to which I wish to call attention, the following experiment was made:

Gold and silver were thoroughly incorporated with mercury, forming a moderately stiff amalgam. This was pressed between two glass plates, and the portion that oozed out in drops was separated from the dry residue. Each part was melted up with lead and assayed for gold and silver. The more fluid part yielded 9.4 milligrammes of gold and 164.4 of silver. The residue yielded 7.8 milligrammes of gold and 159.75 of silver. In other words, the ratio of gold to silver in the more liquid amalgam was as 1 to 17.5, while in the residue it was as 1 to 20.5.

These facts have, I think, a bearing upon the phenomena of scale-formation which have been discussed. When a free-milling ore passes over an amalgamated plate, the gold and silver, and small quantities of other metals, are caught by the mercury. In most cases the process of cleaning-up removes the resulting mixture. Occasionally, however, due perhaps to less fluidity of the amalgam, the pressure of the scraper effects a partial separation, in a manner similar to that of the experiment cited. The mercury squeezed out has probably always a tendency to carry with it fixed relative proportions of gold and silver, as influenced by chemical affinity, but various other causes greatly modify this result.

I think there can be little doubt that silver has in general less tendency to amalgamate than gold. The increase in the proportion of silver at a distance from the battery-discharge, which means that more gold relatively is caught at first, supports this proposition. At the Drum Lummon mill the clean-up

bullion had the ratio of silver to gold of 1 to 1.25, while the ratio in the scale was 1 to 0.76; at the El Silencio mill the ratio in the clean-up was 1 to 1.71, and in the scale 1 to 1.50. These figures also apparently show a tendency of the gold to follow the mercury.

The Physics of Cast-Iron.

Continued Discussion. (See vols. xxv., 84, 964; xxvi., 176, 997.)

(Atlantic City Meeting, February, 1898.)*

THE SECRETARY: The work of Mr. A. E. Outerbridge, Jr., of Philadelphia, first made known in his paper on "The Mobility of Molecules of Cast-Iron," read at the Pittsburgh Meeting of the Institute, in February, 1896, has been reviewed by the Committee on Science and the Arts of the Franklin Institute, whose report, dated May 5, 1897, is accompanied with a table of tests made by the Committee, and amply confirming the statements of Mr. Outerbridge. Concerning the originality and novelty of his investigation, the Committee says:

"The fact that a certain molecular change takes place in cast-iron in the course of time has been known and recognized by many persons for a long time, and that this change tends to relieve the internal strains and render the casting tougher has often been noticed and acted on, although in a very crude way; but, so far as the sub-committee has been able to learn, Mr. Outerbridge has been the first to make a careful study of this, and to note that the change may be hastened by impact, and the whole effect produced in a short time, and also to make a scientific study of the phenomenon. We believe great credit is due him for the initial observation and the careful work dependent thereon. The results were first given to the public in a paper read before the American Institute of Mining Engineers, on February 20, 1896. This aroused a wide-spread interest and discussion among users of cast-iron, and so far as the sub-committee has been able to determine, it has not been shown that anyone, either in this country or abroad, has antedated Mr. Outerbridge in his work in this line. The Franklin Institute therefore recommends the award to him of the John Scott Legacy Premium and Medal."

This report has not been published, but I have had the opportunity to read it in manuscript. The table of tests accompanying it comprises 91 tests of "rumbled" and an equal number of "not rumbled" companion-bars of cast-iron, 26 by 2 by

* See Preface.

1 inch in size and tested flat, 24 inches between supports. The bars were furnished by four different foundries.

One of these tests (No. 63 in the table) shows an increase, due to rumbling, of 42 per cent. in strength and 48 per cent. in deflection—more than twice the maximum gain shown in Mr. Outerbridge's original paper, two years ago. Another test (No. 42) shows 55 per cent. increase in deflection. In several instances (Nos. 14, 29, 40 and 44) defective bars, rumbled, were stronger than their companion-bars, not rumbled, though the latter were free from defects.

Mr. Outerbridge writes me that he has been informed that recent specifications for castings made of strong iron contain a clause to the effect that the test-bars must not be rumbled or otherwise subjected to prolonged vibrations.

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Transactions

vol. 27, 1897

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